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The Position of Electrochemistry in the Chemical and Metallurgical Industries.

When the story of the industrial evolution of the United States will be written by some future historian who must be both a philosopher and an engineer, one of his brightest chapters will deal with the gloriuos history of electrochemistry at Niagara Falls. On Aug. 26, 1895, the Niagara works of the Pittsburgh Reduction Company started the manufacture of aluminium. On Oct. 19 of the same year the current was turned on at the works of the Carborundum Company. In quick succession came calcium carbide, artificial graphite, phosphorus, ferro-alloys, made in the electric furnace, and metallic sodium, caustic and chlorine, chlorates, made by electrolysis. In the span of 15 years these pioneer electrochemical industries and others that followed have become so firmly established in the industrial world that their products are well-nigh indispensable in present civilization. We have forgotten that not so long ago aluminium, carborundum, calcium carbide were unknown. The revolutionary features of the early inventions are gone. What remains are splendidly flourishing industries developing along logical lines into ever-extending fields of usefulness. We have almost learned to look at the pioneer Niagara works as the older generation of electrochemical industries. This is said in a spirit of filial pride. For if Germany may be justly proud of having nursed and raised the coal-tar chemical industries, the United States can claim applied electrochemistry as distinctly American. The bulk of these electrochemical industries was not imported, it was made, born, nursed and raised at Niagara by such pioneers as Acheson and Hall and by those who worked with them or followed them. So far they represent the most distinctly American phase in the history of applied chemistry. But from a broader viewpoint—that of the conservation of our natural resources about which many talk so much and think so little—electrochemists have done more at Niagara. They have made at least a start in modifying popular conception of beauty in the direction expressed in Lord Kelvin's remark that "the great cataract will never be beautiful until it has ceased to be such an awful and unlovely example of waste." There is dramatic beauty in the kinetic energy of the Falls, but there must be and will be as much beauty in the transformation of this kinetic energy into the potential energy of innumerable electrochemical products for distribution for the welfare of mankind all over the land.

Aside from these specifically electrochemical industries, electrochemical methods are exerting an ever-increasing and broadening influence in general metallurgical and chemical industries. Of course, there is nothing essentially new in this development. Electroplating is one of the oldest electrical arts, and electrolytic copper refining, carried out on a large scale since the start of the Balbach refinery in 1883, has now assumed enormous proportions and has become indispensable in modern copper metallurgy, both on account of the necessity of pro-

ducing very pure copper for electrical work and of the easy recovery of the precious metals. But, after all, these were applications of electrochemistry to isolated though important problems. The more general tendency towards electrochemical methods did not manifest itself until the success of the aluminium, carborundum, calcium carbide and graphite industries had attracted general attention. To some extent it is the relation of effect and cause. But that is a somewhat superficial view. It cannot be merely incidental that practically simultaneously theoretical electrochemistry came into existence by the foundation of the electrolytic dissociation theory through physicists rather than chemists in Germany and the large electrochemical industries were founded through American engineers rather than chemists. Clearly the scientific as well as the industrial atmosphere was supersaturated and those two nuclei were the first to form and to grow cystallike. Naturally, others followed quickly. So there came into existence the tendency to try and apply electrochemical methods to all possible problems of general chemical and metallurgical engineering. The immense possibilities of industrial electrochemistry became apparent as well as its limitations. Electrolytic refining was extended to silver, gold, and lead. In gold metallurgy there are now on one hand the various attempts to improve the efficiency of the amalgamating plate by making it a cathode in an electrolytic cell and, on the other hand, the proposition to substitute cyanamide with electrolysis for cyanide. In zinc there is great activity on one hand in the development of electric furnace processes, and, on the other, in dry chlorination and electrolysis. In the iron and steel industry the peculiar local conditions of California, Canada, Sweden and Norway have encouraged large-scale and well designed tests of reducing pig iron in the electric furnace; but the largest and most important industrial development is the appearance of the electric furnace for steel melting and refining on a commercial scale. In chemistry the electrochemical fixation of atmospheric nitrogen in form of nitric acid and nitrates or in form of calcium cyanamide affects the explosive and fertilizer industries. Thus there is rapid evolution. The fact that it is still rapid in many lines proves that equilibrium has not been reached, that we are still far from finality. But this evolution is concentrated on distinct promising problems and proceeds along distinct logical lines. The most characteristic feature is perhaps the broadening of the viewpoint from electrochemistry to physical chemistry. Thus in the fixation of atmospheric nitrogen we have in cyanamide manufacture the problem of separating nitrogen from oxygen by cold, in Haber's ammonia synthesis the increase of reaction velocity by certain catalysers with unusually high atmospheric pressures. Thus this evolution has had an immensely broadening effect on all the workers. The rise of the chemist above the pure analyst, the emancipation of the chemical engineer, the dissociation of the metallurgical from the mining engineer are logical concomitant phenomena; and chemical and metallurgical engineering becomes essentially applied physical chemistry.

The Metal Markets in Retrospect and Prospect

It can be well said that just at present the metal markets are slowly recovering from a neurasthenic condition. Now, excessive introspection is equally bad for anybody in such a state. It is usually better to work than to think about the "whys" and "wherefores." Yet, "if we must philosophize, we must philosophize, and if we must not philosophize we must still philosophize." Accordingly, with the New Year's chimes of Trinity Church still ringing in our ears it may not be amiss to look over the metal markets of the past year and also to see what are the signs for the future. Considering the metallurgical industry from the standpoint of a few years back we see that the great increase in production and its concomitant consumption starting in 1898 continued with more or less momentum for 10 years, until the panic of 1907 and the industrial depression that followed it put effectively the brakes on the movement of expansion; that subsequent to this the metal industries recovered very quickly in tone, until the summer of last year, when a second retardation was felt, followed now by a period of liquidation, quiescence and perplexity.

The reasons for this are many—for, indeed, in what dominion of human knowledge is there not a multiplicity of causes? But mainly and chiefly the present unpleasant condition can be attributed to the fact that we recovered entirely too quickly from the period of retrenchment and liquidation of 1907, starting too soon on an error of extravagance and inflation. Secondarily, we have as minor causes the fear of more legislation that is hostile in intent to capital and the uncertainty about the judicial inquiries into the commercial ethics of the trust question and the expounding by the Supreme Court of the eternal verities of the law to changed conditions. Naturally, this has induced a spirit of unrest in the investing public. Direct in its application to the metal markets is the refusal of the railroads to buy in their usual quantities. In quite a human way the railroads do not care to purchase unless they are allowed to raise freight and passenger rates as compensation for the increased wages given their employees. Furthermore, we have the everywhere-present demands of the wage-earning classes for continual increase of pay. In one intense aspect, we have a strong inhibiting cause. That is the high price and uncertain market for cotton, the staple which means so much to us as an exporting nation and so much more to us as a manufacturing nation. All the above enumerated causes act as mental depressants on all industry and though exaggerated in point of fact are none the less potent in their effect on the minds of business men. In short, things are unsettled and no sure foundation for such an expanding industry, such as normally is the metal and chemical industry, can be made until things are more definite.

Let us consider briefly in this light the several special markets aside from the iron and steel market, which is discussed in another column. Like our steel men the copper producers have had also their pleasing prandial conferences. Possibly the dire straits of some of these convivial spirits has moved to compassion the industry and helped to hold up the price of the red metal. At any rate, production has been curtailed. The electrification of the railroads is proceeding much slower than had been anticipated by the large copper interests. The development of the large porphyry ore beds of Utah, Nevada, New Mexico and Arizona has led to the existence of mines that can make copper at an extremely low price. The exactitude with which the cost of development and plant and the cost of producing the metal can be prognosticated has attracted capital rapidly to this kind of mines. The net result is a largely increased and still increasing productive capacity which can sell copper at a profit at 10 cents a pound.

This has effected an over-production and we have seen the price of copper held down to an average of less than 13 cents with a large surplus built up in years of industrial depression. We are not so bearish on the copper industry, especially as enforced economies are reducing the production costs of the older producers. Besides, the low price has caused the use of copper in many new minor ways in the building trade. In this age of electrical development there cannot long be too much copper. It is interesting to note that the high prices of copper in 1906 and in the first half of 1907, due quite largely to the short-sighted policy of the producers in exaggerating the runaway market for the metal, is a cause of this overproduction. For this high price stimulated the promotion and development of these large low-cost producers that now threaten the market. "As ye sow, so shall ye reap." In the steel industry a broad policy has certainly been pursued, even if cynical Wall Street sees nothing but glittering generalities in the pleasant dinners of the gentlemen directing the policy of the different steel concerns. If a similarly broad policy like that observed in the steel industry had been pursued with copper things would have been better for the producer to-day. It is not unnatural to expect in the course of time the consolidation of the several larger concerns into one concern or trust that will control 50 per cent of total production. This will probably be better than the proposed large central selling agency. But the psychological moment for the birth of the copper trust has hardly yet arrived. A logical inference is that the definite character of the ore reserves of the porphyries will lend the stability to the concern that is needed to enlist the support of the banking interests. It would almost seem that the conditions now are exactly analogous to those existing 11 years ago when the great iron ore beds of the Mesaba range assumed an increasing industrial importance.

Turning to the zinc business, we find that it has only been satisfactory to the mining end of the business. The market for spelter has fluctuated between fairly high limits, but the smelting margin has been so low that money alone was made by the smelting concerns possessing great power and hold on the market. The final decease of Iola as a smelting center after years of a prolonged lingering illness is expected this winter. The tendency is to build new spelter plants in the Illinois coal fields. The sensational finds of carbonate ore in Leadville and the reduction to a commercial basis of the Huff electrostatic separator will, among other factors, ease in the ensuing year the position of the smelters. While it is so cheap to build competitive plants, and small plants can be so efficiently operated in the zinc industry, the idea of a zinc trust is chimerical.

In the lead market restricted production of ore, due to the approaching exhaustion of certain large mines, has unquestionably brought the market into a position much brighter from the viewpoint of the producer. Statistical evidence is wanting, but it is probable that the price for 1911 will average higher than the price for 1910.

Silver has been absorbed by the Far East to quite an extent. There has been of late a generally healthy tone to the silver market and a continuation of this can be legitimately anticipated. It is the one metal save gold—for which, as many an alluring gold mine prospectus has told us, Uncle Sam is an unlimited customer—about which there is little perplexity.

In all the other markets the chief feature is the perplexity and doubt on the part of the consuming public. Could an intelligent despot by his fiat take the stand and say "all metals will be sold as per a certain schedule for a period of five years," and could he make the statement with means to back it up: "I will buy copper in any amount for 13⅞ cents and I will sell copper in any amount at 14 cents," consumption would increase at once. But fortunately no one man possesses the power or the means or the intelligence needed to be such a despot. And the only cause for the perplexed human consumer is to pursue a careful course.

In conclusion, we might say that metals are produced now more efficiently than ever before and that each year civilization finds new uses for them. This is a general encouraging feature. In particular a bright feature is that our crops are unrivaled in size and prices lower. Of course, we will always have social agitation, for the extravagance of the foolish must always be expected and a certain class of people will be discontented and agitate for their alleged rights. Questions that perplex the metal markets to-day will not perplex them next year. Then there will be a new set of troubles. We do not now think that the metal industry is in such a bad way. For when things are bad, they are never really as bad as people imagine. Just one thing is certain. Those in the metal business who work hard and intelligently at their own affairs will have little time for morbid introspection and what philosophizing they do will be pleasant.

Iron and Steel in 1910.

The story of what occurred in 1910 in the iron and steel trade is soon told. Why it occurred is, as our friend Kipling would tell us, "another story," and, we might add, a much longer story, if, indeed, it can be told at all. The year opened with the United States producing at the rate of about 31,000,000 tons of pig iron a year, and finished products in proportion. Three years, 1906, 1907 and 1909, had shown a production between 25,000,000 tons and 26,000,000 tons, but no calendar year had crossed the 26,000,000-ton mark. From April, 1909, to February, 1910, there was a steady gain in production. After February production declined, somewhat rapidly during the second quarter, very slowly during the third quarter, and very rapidly during the fourth, so that the rate at the end of the year was not much above 20,000,000 tons, in terms of pig iron.

Prices of pig iron declined almost steadily, but very slowly, through the year. Prices of finished steel products declined rather easily from early in January until the forepart of August, total declines in finished products outside of rails and tin plates averaging not far from $3 a ton. The prices of rails and tin plates did not decline during the year. In August the nominal prices were shaded in some cases, whereas at the close of the year the same nominal prices prevailed and were not being shaded. The circumstantial evidence is that a "controlled market" came into being.

The steel mills entered the year with a very large, though not unprecedented, volume of business on their books. Uniformly shipments exceeded new bookings, and month by month the accumulation decreased until by the close of the year the

mills were shipping almost wholly on current orders. A natural sequence to the continued expiration of old contracts was an increase, beginning in August, in new bookings, but as the new buyers thus appearing were merely those whose contracts had expired the actual demand for material did not increase thereby.

We estimate the production of pig iron in 1910 at 27,150,000 ± 150,000 gross tons, comparing with previous years as follows:

1902, 17,821,307	1906, 25,307,191
1903, 18,009,252	1907, 25,781,361
1904, 16,497,033	1908, 15,936,018
1905, 22,992,380	1909, 25,795,471

1910, 27,150,000 ± 150,000.

The calendar year is altogether too long a period to show the real vicissitudes of pig iron production. While neither 1906 nor 1907 showed as much as 26,000,000 tons of pig iron produced, the twelvemonth ended June 30, 1907, showed 26,200,000 tons. The calendar year 1908 fell fairly well in the trough of reduced production, but 1909 and 1910 fell one upon the ascent and the other upon the descent of the curve. Maximum production was in February, 1910, and the twelvemonth of greatest production was that ending July 31, 1910, it covering a production of no less than 29,800,000 tons.

The explanation of why a year opening with such promise broke previous production records and yet constituted on the whole a bitter disappointment would be written differently by different observers, according to their viewpoint. One, full of old-time ideas of the importance of railroad buying, would attribute the recession in buying to the refusal of the government to allow the railroads to advance their freight rates, at least without the serious consideration of the Interstate Commerce Commission, which is still in progress. It was early in April that it became apparent that the rate advances could not be made without a contest, and about that time railroad orders became light. Another, accustomed to regard political developments as exerting one of the most serious influences upon business, would cite the change in political sentiment in the country as the cause of the slowing down in the industry. A third, irrevocably given to the open-market method of transacting business, would ascribe the reversal in the iron industry to the recently devised system of price control, being responsible, first, for an unhealthy boom after the great price break of February, 1909, following a period of 15 months' price control during a period of slack demand, and, second, for the abstention of buyers in the closing months of 1910 when price control was again in evidence.

Probably all these causes contributed toward making 1910 what it was. In the upward movement in the second half of 1909, after prices had found the lowest level for nine years, producers frankly stated they were advancing prices to choke off buying, but the more they asserted that the more buyers were encouraged to buy, in some cases making contracts for forward delivery, in other cases actually taking out the material and adding to their stocks. To support the position in which the iron and steel industry found itself at the opening of 1910 there was required a general and marked improvement in the general consumptive capacity of the country. This was experienced, perhaps, to a limited extent during the early months of the year, but when railroad buying was largely shut off the industry had to run on its accumulation of railroad orders. The accumulation decreased rapidly toward the close of the year, while the general requirements of the country probably underwent some diminution also.

At the close of the year the paramount question was: Is the liquidation still running its course, or, as the revival a year earlier went too far, has the recession also gone too far? A very few months of the new year will determine, and we think it will determine in favor of the latter alternative. To the close-range view the matter of prices of finished steel products is an element, there being a sentiment widely prevalent among buyers and not absent entirely from the ranks of producers that finished steel prices should be reduced. This we consider a detail which the more vital elements will settle. If business is in position to revive in a marked way, the present system of price control will yield. If there is no hope for radical improvement in any case, or if there is good promise of improvement without price reduction, prices will hold.

Western Steam Coal.

Chemists and engineers long have recognized important differences between Eastern and Western coals. These differences are so marked that methods of proximate analysis devised for Eastern coal are not always suited to Western coal, or at least the results of such analysis on Western coal are not capable of the same interpretation. Again, methods of boiler firing and types of boiler construction which are useful and efficient elsewhere have been found deficient in the West. In many sections of the West the coal available for steam purposes is a lignite containing 40 per cent to 48 per cent fixed carbon, 32 per cent to 37 per cent volatile matter, 5 per cent to 23 per cent moisture and 2 per cent to 14 per cent ash. The calorific value is about 11,000 b.t.u. The high percentage of volatile matter is the distinctive feature of this coal. If consumed under ordinary conditions, this same volatile matter, which contains about 17 per cent of the calorific value, is also the cause of the greatest loss in efficiency, so that even though the coal may have the same calorific value as an Eastern coal it will give much lower efficiency when burned under the same conditions. This is due to the fact that the volatile matter is easily driven off from a freshly fired charge of coal at a temperature low enough to distil it, but not sufficiently high to bring about combustion. The gases thus escape from the zone of action unburned and reduce the efficiency of the operation.

These peculiar conditions have caused Western engineers to devise special furnaces and means of firing, whereby the efficiency of Western coal may be greatly increased over former practice, and made to approximate closely that of the East. A thick fuel bed, mechanical stoking and forced draft, with preheated secondary air, have been found valuable in burning this fuel. This idea has been applied not only to boilers for steam-raising purposes, but also to metallurgical furnaces, such as roasters and reverberatory smelters. Forced draft has been applied in various ways and improvement has been marked in this line. Using steam direct for forcing the draft has consumed about 18 per cent of the steam raised. When using fans driven by steam engines the steam consumption has been reduced to about 12 per cent, while more recent devices embodying small turbines and fans are credited with a consumption of

only 4 per cent to 6 per cent of the steam raised. A difference of opinion exists as to the value of injecting steam with the draft, although this probably refers more to the quantity used and manner of application than to its total elimination. Another interesting phase of the use of Western coal is the improvement which is noted where coal is bought on specification, with a graded schedule of prices for fuel value. One notable instance is that of an institution using slack of a certain specification. As long as little attention was paid to the grade of the coal it was of poor quality and the efficiency of the steam plant was low. As soon, however, as chemical control was instituted, the grade of the fuel promptly rose with corresponding efficiency in the power plant.

National Faults in Manufacturing.

The American people have been styled often by Europeans "a nation of shopkeepers." This remark certainly has a modicum of truth in it. While it is perfectly true that we display at times an intense striving for the immaterial and for the spiritual, yet all of us are more or less of the time quite nimble dollar-chasers. Possibly in our desire for acquiring money we overreach the mark, but we are not the only ones. American visitors in England and continental Europe have remarked on a certain convulsive grasping after the last farthing, centime, or pfennig that suggests that Europeans criticism of our weakness for money is founded on the critics' ability to see the mote in the American eye and to disregard calmly at the same time the Continental beam.

However that may be, concisely stated, Americans are business people and are fond of the business game. Now this tendency is often too intense and the vision focused so closely and so continuously on the intermediate object in view as to make the American business man myopic. A quite prominent manufacturer of special steels was discussing recently an intricate phase of the working of the automobile with a metallurgical engineer in a New York club. The former showed such an intellectual grasp of the broad principles of a business foreign to his own that the latter ventured to ask him whether he was considering the introduction of the electric-steel furnace in one of his plants. Quite to the surprise of the metallurgist this man, broadly intelligent outside of his own field, was only narrowly intelligent in his own field. He confessed never to have heard of the electric-steel furnace as being anything more than a toy. Now, for years this journal has been reciting the potential advantages of electric steel making and also the practical achievements of the several electric-steel furnaces in this country and in Europe. Recently also the general technical press has not been backward in following the lead of METALLURGICAL AND CHEMICAL ENGINEERING. Furthermore, the largest steel company in the world has demonstrated the commercial utility of the electric-steel furnace. And yet this manufacturer of special steels, so keen in looking at the automobile business in a philosophical manner, had centered his mind so closely to the actual carrying out of his operations from day to day that he could not see that his concern would be saved from the ever vexatious labor question by the introduction of the electric-steel furnace. Verily, as the sage remarked, "some people cannot see more than six inches beyond their nose."

Our metallurgical friend had the tact to awaken the manufacturer slowly and also the sagacity to test the sincerity of his remarks. For some people assume an attitude of ignorance in order to gain expert knowledge without paying for it. The steel manufacturer simply had the inability to see the architectonics of his manufacturing structure. He had exaggerated the value of salesmanship (that side of the subject he understood perfectly) and he laid too much stress on the value of practical and resourceful foremen. These details are, of course, well-nigh essential and always exceedingly important. But, but, and but, they are not the sum total of high-class manufacturing.

The three prime elements of successful manufacturing are, first, to have a clear view of the commercial and financial aspects of the situation; secondly, to pay close attention to the practical and engineering sides of the subject; and thirdly, to have ever in mind an imaginative, yet reason-tempered conception of the many possible applications of physics, chemistry, and other pure sciences to the special class of manufacturing on hand. This analysis is but a rough one, just as an analysis of concrete as an aggregate of cement, sand and crushed stone without further chemical or physical determinations is a rough, general analysis. But though rough, we believe it sufficient and practical for pragmatic purposes. It is accordingly only too evident that the American manufacturers exaggerate the importance of the engineering and strictly commercial sides of business affairs.

To attain perfect sanity in an individual a proper balance of the physical and mental and emotional energies is necessary. Without these co-ordinated properly, a departure from the highest normality is made, varying, according to the disharmony of the forces and their magnitude, from everyday folly and extraordinary silliness, till eccentricity, incompetence and finally insanity is reached. Just so in the practical world of business to attain the fullest measure of success there must be a co-ordinated balance of the three different components. The average American manufacturer has overdeveloped two sides and neglected the third side. When he turns to the use of the projected imagination he often shows a queer form of activity as, for instance, the desire to get gold out of seawater, or the willingness of a banker to capitalize an attempt of making silver from lead. Probably his imagination is so seldom used that when it is used it is stimulated to such an extraordinary degree that his otherwise sound judgment is paralyzed.

While American manufacturers do not pay attention enough to the possibilities of applied science, German manufacturers perhaps pay too much attention to pure science. In many large German works a few less doctors of philosophy and a few more hard-headed Yankees would improve things. It is not necessary for us to criticise much the policy of our Teutonic confrères. They have developed their business to meet the actual conditions confronting them. But they do not know all the possibilities of the game any more than do our own manufacturers. Both have characteristic points of weakness. Each country can learn and, we are glad to say, is learning from the other. For there is no tariff on the interchange of ideas. It is grandly true that in the realm of ideas reciprocity expands freely by laws made not by man but by God.

Perkin Medal

At a meeting of the Perkin Medal Committee, consisting of representatives of the Society of Chemical Industry, the American Chemical Society and the American Electrochemical Society, the Perkin medal for the present year was awarded to Mr. Chas. M. Hall, of Niagara Falls, for his pioneer work in the aluminium industry.

The formal presentation of the Perkin medal to Mr. Hall will be made at the next meeting of the New York Section of the Society of Chemical Industry.

The Western Metallurgical Field

British Columbia.

The investigation of the zinc resources of this province, which began about five years ago, is still being prosecuted by the Dominion government under the direction of Dr. Eugene Haanel, director of the Department of Mines. Experiments are now being conducted at McGill University for the purpose of solving some of the fundamental difficulties in the reduction of low-grade zinc ores by electric methods. When this is accomplished it is proposed to erect an experimental plant at Nelson, where tests will be conducted on a larger scale. Electric smelting is only one of the processes being investigated by the commission having the matter in charge. Ore has been shipped to the university from a number of the properties having zinc ore, and the possibilities of treatment are being thoroughly investigated by the commission, which includes Dr. A. Stansfield and Mr. W. R. Ingalls.

The new Portland Canal district was recently visited by the provincial mineralogist for the purpose of issuing a preliminary official report. As yet there is but one producing property, although a number of owners are prosecuting development work. The Portland Canal Mining Company, of Victoria, has erected a 75-ton concentrator, which is designed to be one unit of a larger mill if the developments warrant. Ore is conveyed by aerial tramway, and concentrated about 4 into 1, with a clean separation and good saving.

Alaska.

Sluicing and dredging in the Nome district were brought to a close earlier last year than usual, and as a result of this and other causes the output for 1910 will be from $250,000 to $500,000 less than for 1909. The production from the Seward Peninsula as shown by the Customs Office was about $3,500,000 for 1910, as compared with slightly over $4,000,000 for 1909.

Other conditions contributing to the reduced output were the early closing of the navigation season, a late spring, and the partial exhaustion of some of the pay streaks. Nineteen dredges were in operation during the year, but of this number 16 closed unusually early.

About 10 new dredges were built in the peninsula last season, and these, together with others to be built next year, will make a total of 24, most of which will commence operations as soon as possible next spring.

Mexico.

The recent demonstration against the government has not amounted to as much as United States papers intimated, and personal advices from the southern republic indicate that the revolution has never been serious. In view of the large metallurgical interests controlled by American capital in Mexico, it is gratifying to know that there is no immediate prospect of a disturbance there. There can be no doubt that the growing supremacy of American capital and industry in Mexico is distasteful to a part of the Mexican population. On the other hand, the officials have recognized the value of the foreign element and have encouraged their work by offering favorable concessions.

The Pacific Smelting & Mining Company, a New York concern, is laying plans to become a strong custom smelting company in northern Mexico. It has recently purchased a concession for erection and operation of a custom smelter at Mazatlan, Sinaloa. Two other plants are under construction by this company, viz., one each at Fundicion and Guaymas, Sonora, and both will be in operation early this year. With three plants drawing from the mining districts of Sonora, Jalisco, Tepic, Sinaloa and Durango the company will be a strong factor in Mexican metallurgy.

The Cananea Consolidated Copper Company, of Cananea, Sonora, will treat the concentrates produced by the new mill of the Miami Copper Company, Arizona, delivery to begin about January, 1911. Special bin and furnace equipment have been installed to accommodate this increased supply of ore.

Nevada.

One of the less prominent districts of Nevada is known as Seven Troughs. Although not so well known as some others, it produces some rich ore and has some mills that are reported to be doing good work. The Seven Troughs Coalition Mines Company has a 10-stamp amalgamation-concentration mill in which the low-grade ore is treated with a good recovery. The tailing loss is expected to be recovered by cyanidation in the future, and tailings are being impounded in anticipation of installing that process.

The most pretentious plant in the district is that of the Darby Ore Reduction Company, at Mazuma. Ten stamps reduce the ore to pulp, which then passes over plates, and later, over Deister concentrating tables. The table tailings are classified into sand and slime by Dorr classifier, the sand being ground in tube mill and amalgamated before joining the slime for cyanide treatment. The latter process is carried out by agitation in tanks and filtration on Oliver continuous filters.

One of the newest districts in Nevada is known as Jarbidge, situated in the northern part of the State. The developments have proceeded far enough to warrant one company, the Pavlak, erecting a cyanide mill, which is now under construction. In order to aid the pioneer companies as much as possible the county commissioners have ordered a road built into the camp, making good connections with the Western Pacific Railway at Deeth.

Utah.

The recent report of the Utah Consolidated Company, which showed such marked discrepancies between the supposed and actual state of the property, was a great surprise. It had been generally supposed that, while the Highland Boy mine was not in the promising condition which it once enjoyed, it was, nevertheless, in possession of three to five times the reserve which the last report showed. This opinion was supported not only by the previous reports on the property made less than a year ago, but by the fact that the International smelter was built at Tooele ostensibly relying on the ore from the Utah Consolidated for its main supply. At present it appears that the anticipated ore supply will not be forthcoming, and that another year will see the finish of shipments from the Utah Consolidated to the International.

This has necessitated a campaign for other ore supply, and one of the possibilities mentioned is that the Giroux company will ship to Tooele instead of building its own smelter in Nevada. This would involve an adjustment of freight rates over several railroads, but in view of the difficulty in securing a large copper tonnage under present conditions, shipments might be made even if arrangements were not as good as the best. The Tooele plant is our newest and most complete copper smelter, as has been outlined previously in these columns, and favorable rates would undoubtedly be made rather than have it idle wholly or in part.

In the meantime conditions at the Garfield plant of the American Smelters Securities Company are more flourishing than before, both as to capacity and ore supply. A sixth reverberatory furnace has been built, and the converting capacity has been increased by additional Peirce-Smith basic lined converters. All of the reverberatories and three out of four

blast furnaces are running. These changes have been made necessary by the increased output of the mills of the Utah Copper Company at Garfield, and the combination bids fair to make Garfield the greatest copper concentrating and smelter center in the country.

Experiments looking toward improved zinc concentration at the Daly-Judge mill, at Park City, have been under way for some time. Retreatment of the mill slime on tables is being tested, and in addition to this tests are being made with the MacQuisten tube in an endeavor to improve the quality of the zinc product.

Colorado.

The recent completion of the Newhouse tunnel in the Gilpin district, and the connection of the Roosevelt tunnel at Cripple Creek with the El Paso shaft, are regarded as two events which will have an important bearing on the immediate future of Colorado mining and metallurgy. The Newhouse tunnel is 21,968 ft. long, and provides drainage and transportation facilities for a mineralized area which has been very productive and which still contains large quantities of ore suitable for preliminary concentration or final reduction in the mills of the district. The Roosevelt tunnel has been driven about 16,000 ft. and has 14,000 ft. yet to be driven before its objective point is reached. In the latter portion of its course it will effect the drainage of the area which has a common water level and which contains numerous gold-bearing veins. In view of the undoubted efficacy of the cyanide process in the treatment of these ores the completion of the tunnel will undoubtedly open the way for the erection of new mills and the creation of a new metallurgical era for Cripple Creek.

The Non-Ferrous Metal Market.

A general state of depression has pervaded the market for non-ferrous metals, and the feeling regarding the future is divided. Interest in copper is centering about the efforts being made to realize an agreement among the large producers, for it is believed that some plan of co-operation will be necessary for the good of the industry. Prices for all metals have been almost stationary or reduced since our last monthly report.

Copper.—Although the producers are endeavoring to hold the market at the advance which it experienced in November it is apparent that purchasers are not inclined to buy at the high figure, and as a consequence the prices are being shaded for most transactions. Lake copper has been offered freely at 12¾ cents to 12⅞ cents, and electrolytic at 12.5 cents to 12.6 cents.

Lead.—Considerable business has been transacted on contracts, but the market is quiet. New York quotations are 4.5 cents, and St. Louis 4.35 cents to 4.37½ cents.

Tin.—The advance in the foreign market and the apparent control of that market by the bull interests has resulted in a slight advance at home, although the volume of business is light.

December tin was quoted in New York at 38½ cents.

Spelter.—In common with the other metals, spelter has been depressed and has lost its favorable position reported last month. Inquiries for metal are few and competition among the sellers has been keen. Prices have steadily declined from week to week, and the metal is quoted at 5.47½ cents to 5.52½ cents, St. Louis, and 5.62½ cents to 5.67½ cents, New York.

Other Metals.—Aluminium has remained quiet at 22½ cents to 23 cents for No. 1 ingots. Antimony is unchanged at 7 cents to 7⅝ cents for various brands. There is a fair business in quicksilver at $42 per 75-lb. flask at New York and $41.50 at San Francisco.

Camp Bird, Ltd., Ouray, Colo., operated its mill 30 days in September last, treating 6690 tons of ore and recovering 6281 oz. of bullion and approximately 761 tons of concentrates. The monthly report for September shows about $85,703 applicable to profit.

The Iron and Steel Market.

The stagnation which developed in the iron and steel market late in November was intensified in December, and further and sharp reductions in output were made in the steel industry. In the merchant furnace trade some curtailment of output occurred, but obviously not enough to take up the slack, and December witnessed an increase in the accumulation of pig iron stocks which has been practically continuous all year.

Following the general meeting of steel manufacturers in New York on Nov. 30, a series of meetings of producers in individual lines occurred, the meetings covering in turn rails, structural shapes, sheets, tin plates, merchant bars and plates. These meetings were held not for the purpose of fostering a definite price agreement, but rather for the purpose of interchanging information and encouraging manufacturers to curtail output.

Many plants closed entirely for the last fortnight in the year, and others closed for the last week. Altogether there has been the greatest holiday slowing down in the industry that has occurred for many years. At the close of 1903 and again at the close of 1907 there was sharp restriction of output, but in those cases the definite movement began in October or earlier, while last month the curtailment was so great, following the very gradual curtailing tendency which had been in evidence, that it partook of the nature of a separate movement.

Prices of finished steel products have been very steady, there being less shading of openly recognized prices, as a rule, than in October or November. The precise cause of the steadiness cannot be stated absolutely, it being apparently the resultant of the meetings which had been held in the trade and the complete indifference of buyers, which removed quite largely the temptation to invite business by cutting prices.

In the first week or two of December production of pig iron was at the rate of about 22,500,000 tons annually, contrasting with a maximum rate of 31,600,000 tons, reached in February last. Considering, however, that the merchant furnaces curtailed their output much less than the steel interests, it may be estimated that the steel industry in the forepart of December was running at only two-thirds the February gait. Toward the close of December the reduction was so sharp that no precise estimate can be made. The prospects of resumption are uncertain. Some plants are definitely scheduled to resume operations the first week in January, but in other cases there will probably be a wait for some indications of the character of business in the new year.

Among buyers the impression is widespread that finished steel prices will be lower within a few weeks, either through an orderly reduction or an open break. It has been generally reported that the chief desire of the steel interests was to hold prices until after the turn of the year, so that if lower prices were to be sanctioned the point could be reached by an impressive and formal reduction.

Pig Iron.

The market has continued quiet, without any important changes in prices. Demand has been light and it is evident that melters aim to have their obligations at a minimum. Several meetings of blast furnace interests have been held, chiefly at Philadelphia and Cleveland, looking to the exerting of influences for a better market, but no definite conclusions have been reached. There was a particular desire to bring about better conditions in the basic pig iron market in the Central West, but the work so far seems to have been fruitless. Basic iron had, however, firmed up slightly before the movement was definitely proposed, and is quotable at $13.50, valley furnaces, while as low as $13 was done in November. A sale of about 2000 tons of Bessemer was made for December delivery at $15, valley, the regular market since late in August. Foundry iron has gone at $13.75 to $14, valley, and one interest has been ready to do $13.50 on particularly attractive business.

Steel.

The movement for regular prices on billets and sheet bars was in the first flush at the time last report was written, and since then the understanding has had to yield a trifle at two points. At the time of last report Bessemer billets were quoted at $23 and sheet bars at $24.50, both f. o. b. Pittsburg, plus freight from Pittsburg to destination. The idea now is $23 for billets and $24 for sheet bars, at Pittsburg or Youngstown, with freight to destination from the nearer point. A concession of 50 cents in the spread between billets and sheet bars was made and the separate basing point at Youngstown was re-established. However, the open-hearth steel market seems to have firmed up to the Bessemer level, whereas in November some sales were made at less than the going price for Bessemer. Rods are slightly lower, at $27.50, Pittsburg.

Finished Steel.

The following prices, f. o. b. Pittsburg, except where otherwise stated, are being quite generally observed:

Rails, standard sections, 1.25 cents for Bessemer, 1.34 cents for open-hearth, f. o. b. mill, except Colorado.

Plates, 1.40 cents for tank quality.

Shapes, 1.40 cents for beams and channels, 3-in. to 15-in., zees, and angles 2 x 3 and larger.

Steel bars, 1.40 cents, base; iron bars, 1.35 to 1.40 cents, Pittsburg; 1.30 cents, Cleveland; 1.30 to 1.35 cents, Chicago.

Wire nails, $1.70, base, per keg; plain wire, 1.50 cents; painted barb wire, 1.70 cents; galvanized barb wire, 2 cents.

Black sheets, 28-gage, 2.20 cents; galvanized, 3.20 cents; painted corrugated roofing, $1.55 per square; galvanized, $2.75.

Tin plates, $3.60 for 100-lb. cokes.

CORRESPONDENCE

Presidency of American Electrochemical Society.

To the Editor of Metallurgical and Chemical Engineering:

Sir:—In view of the nomination of officers of the American Electrochemical Society for the year 1911, we, the undersigned, suggest for the office of the president Dr. William R. Whitney.

The abilities of Dr. Whitney as an electrochemist are too well known to need to be stated here. His recognition as a chemist is best shown by the fact that he has been elected to the presidency of the American Chemical Society. His able services as a president of a national society have been shown by his conscientious work as president of the American Chemical Society.

He was one of the two prominent candidates for president at the election of the American Electrochemical Society last year, but declined on the ground that he was then president of the Chemical Society; the reason for his declination last year, therefore, now no longer exists.

We feel that with Dr. Whitney as president of the American Electrochemical Society the interests of that society will be in very able and conscientious hands, and that the duties of that office will be well taken care of.

E. G. Acheson, L. H. Baekeland, H. R. Carveth, Francis A. J. FitzGerald, Carl Hering, F. A. Lidbury, E. R. Taylor.

Heat Loss Through Furnace Walls.

To the Editor of Metallurgical and Chemical Engineering:

Sir:—My attention has been called, by Mr. F. T. Snyder, to a typographical error in formula (4), page 218, Vol. XIV of the *Trans.* Amer. Electrochem. So. in my paper on "Heat Conductances Through Walls of Furnaces," and copied with the same error in the advanced copies of my paper on "Method for Determining Thermal Conductivities" read at the recent Chicago meeting, where it is (6) on page 9. The letter "l" occurring twice in that formula should be the numeral "1."

I take this opportunity to also call attention to the fact that the simpler formula (4) in the latter paper given for a sphere, and (2) in the earlier one, is also rigidly correct for any rectangular furnace and not merely for a cubical one.

Philadelphia, Pa. Carl Hering.

Heat Resistance Units.

To the Editor of Metallurgical and Chemical Engineering:

Sir:—In your issue for December appears an article by Mr. Carl Hering on "temperature drop a true measure of thermal resistance," which to my mind is much to the point. Not only are Mr. Hering's proposals logical from a scientific standpoint, but they are directly applicable to everyday practice.

I would propose that instead of using the English inch as the unit cube for thermal resistivity and conductivity the centimeter cube be used. This would give to these specific properties of materials a multiple relation to the corresponding C. G. S. figures. I would further propose that the thermal unit for resistance and conductance be named not thermal ohm and thermal mho, but that as abbreviation of these "thohm" and "mhoth" or "themho" be used instead.

Is it not eminently proper for the American Electrochemical Society to study now the question of nomenclature, as the above, and ratings, etc., and attempt to standardize terms that would be accepted by the profession?

Hartford, Conn. Woolsey McA. Johnson.

Metallurgy of Zinc.

To the Editor of Metallurgical and Chemical Engineering:

Sir:—I have read with interest the editorial in the December number, entitled "Condensation of Zinc Vapor," in which the work of Mr. Hixon, Mr. Johnson and others is spoken of. The general trend of the editorial seems to be an encouragement of improving the metallurgy of zinc, the "unusual" metal. It also infers that the present metallurgy of zinc is primitive and not in keeping with twentieth century methods. This attitude is certainly correct.

The serious error in the editorial, however, is the failure to recognize the advance of the dry-chloridization of complex zinc ores, originally invented in this country by Messrs. Chas. E. Baker and Arthur W. Burwell, prefaced in Norway by Alf-Sindig Larsson, and in England by Swinburne and Ashcroft. The process has been further improved in this country by Mr. Chas. E. Baker and others at various times connected with him, such as Mr. William Koehler, once associated with Dr. Hoepfner, Chas. A. Diehl and J. L. Malm. Mr. Malm is at present manager of a works in Georgetown, Col., known as "The Western Metals Co.," licensed under the Baker-Burwell patents by Mr. Baker, owner of the patent rights for Colorado.

It is not my purpose in this letter to describe the process, as such has already been done, both in your journal and in a paper by Mr. Baker before the American Electrochemical Society. It is my purpose to call your attention and that of your readers to the error in your editorial, that while you congratulate Mr. Hixon and Mr. Johnson you omit the Baker-Burwell process, which, I believe, is much nearer commercial fulfillment than any.

The process is particularly adapted to ores carrying zinc-lead-iron sulphides with small quantities of copper, silver and gold, and recovers all the values. As such ores lie on dumps in the West in unlimited quantities, the value of the process as a factor in the "conservation of our natural resources" is of great magnitude. Either spelter or zinc and zinc-lead pigments may be obtained, as desired.

Chicago, Ill. Howard Baker.

[As the editorial in question was entitled "The Condensation of Zinc Vapor," its scope was clearly restricted to smelting processes. That, even with greatly improved smelting processes, there would remain a distinct profitable field for a chlorination process, when properly reduced to practice, there can be no doubt.—Editor.]

The Training of Chemical Engineers.*

BY M. C. WHITAKER.

Much attention is properly being centered upon the education of men in the scientific development and management of agricultural production, and the Government not only gives direct financial support to a large number of schools for such training, but also maintains a Department of Agriculture, the function of which is largely educational.

The annual production of the chemical industries is almost equal in value to the agricultural production of the country, and it is, therefore, proper to compare the facilities now being provided and the means now being adopted for the education of men to develop and manage the widely diversified units of chemical manufacture with the methods and facilities in other educational fields. Obviously the general educational and technical equipment of men to develop and direct processes or works depending upon the applications of chemistry, and usually involving a knowledge of power and mechanical and electrical machinery, is much greater than the technical equipment required for men in agricultural production. While agricultural education has grown and developed at a rate commensurate with the producing power of that industry, and has been fostered by private, state and federal funds, education in the equally important and essential field of chemical manufacture has been, in some institutions, entirely ignored, in others only indifferently provided, and in no single case supported and equipped on a basis comparable with the wealth-producing power of the industry.

Efficient production and the economic management of our manufacturing plants are essential features to our commercial development, and it is in this field that the greatest results are to be attained in the conservation of our natural resources. The accomplishment of these results obviously depends upon the application of scientific knowledge to the solution of all problems, both great and small, in the development, the direction and the management of our factories.

The chemical industries very properly look to the schools for men qualified to produce these results, to improve their production and manage their works. The best evidence that more and better trained men are needed in the chemical industries is the fact that this Institute of Chemical Engineers, composed of men of experience and men now engaged in the direction of great industries, meets twice a year and devotes a large portion of its time to the study of the problem of how to obtain trained men to assist them in the development and administration of their work.

Every employer of chemical talent knows that at best the young graduate is only a vague "prospect" and that much has been left undone in his training.

The problem of proper training and correct methods of teaching chemical engineering is one which must be carefully analyzed and studied from its very foundation and is not one to be remedied by a few trivial changes in existing curricula —by dropping this and substituting that. The subject must be opened up and examined to its foundation, and if it is found that the present structure is built on sand we should put some "concrete" under it.

Necessarily the problem must be solved by the co-operation of the teachers with the employer of chemically trained men, and the writer, having served a term of practically 10 years in each of the above fields, begs to submit a few observations and conclusions for the consideration of the institute committee.

The chemical industries require for their development and management two classes of men of somewhat different training, natural qualifications and range of service.

First.—Research chemists who are qualified naturally and by training to originate and to develop in new fields. The training of the research man is more advanced, or supplements that of the chemical engineer, and the research results furnish the foundations for the industries which the chemical engineer organizes and administers. In other words, the research chemist works in the new and undeveloped fields of chemical knowledge and produces results calculated to contribute new knowledge and new arts.

Second.—The *chemical engineer* works in the organization, operation and management of existing or proposed processes with a view to building up a successful manufacturing industry. He uses his classified knowledge of chemistry and the allied engineering branches in developing, perfecting, organizing and administering a plant or a process for the production of a marketable and useful product at a profit to the investor.

Such a field as this requires a man of knowledge, originality and resourcefulness. His fundamental training in chemistry, physics, mathematics, etc., must be thorough and must be combined with a natural engineering inclination and an acquired knowledge of engineering methods and appliances. His originality and imaginative capacity seem to me to be essential and a combination result of natural qualifications and training. Resourcefulness, however, comes as a result of experience in application and the ability to adapt a wide range of methods and practices to new conditions. A man with the natural qualifications, supplemented by the proper training, may become a successful chemical engineer.

The success of the research chemist is measured by his scientific achievements, whereas the success of the chemical engineer is measured by his commercial results. While their work bears a certain mutual and cyclic relation it must be apparent that their natural and educational equipment may be entirely different.

The development of specialized training in certain selected lines of manufacture is not a proper function for an engineering school, but should be left to the trade schools or to the industries themselves. Specialized research, on the other hand, should be carefully eliminated from the regular engineering training and provided for in graduate courses.

For convenience in discussion I have divided training subjects for the chemical engineer into three general classes: fundamental, associated and supplementary. No attempt is made to discuss the details of the curriculum, the general purpose being to get broad subdivisions as a starting point. The question of arrangement of subjects into a curriculum and the proper allotment of time for the course depends upon what is finally determined as proper training for the men under discussion:

Chemistry Physics Mathematics Allied subjects	Fundamental training.
Electrical engineering Mechanical engineering Civil engineering General engineering Business economics	Associated training.
Study of the *applications* of the fundamental and associated training in laboratories equipped with the "tools of the trade" and with working plants. Training by "contact" and by "example" in a laboratory managed as an approved business, in the principles and practices of efficient organization and administration.	Supplementary training.

Fundamental Training.

The training in the fundamental subjects need not be materially changed except perhaps more attention should be given

*A paper read before the American Institute of Chemical Engineers at the meeting in New York City on December 7.

to laboratory facilities and to the selection of instructors for the fundamental work. Profound scholarship and a record for original research work are too often made the sole determining factor in the selection of teachers for these subjects.

Research qualifications are of secondary importance in the selection of men to give the fundamental training to our chemical engineers. Such teachers might better be chosen from men physically, temperamentally and educationally qualified to present their subjects clearly, logically and enthusiastically to the student, and to present them in such a way as to quicken and hold his interest.

Too often our undergraduates become muddled, discouraged and even give up a subject because of its dry, disconnected presentation or because the instructor "talks over their heads" or lacks the personality and the power to interest them. Research instructors are often permitted to emphasize their "hobbies" or teach their special experiences without regard to the purpose of the course and its relation to the fundamental training. Our instructors should always be chosen to fit the course requirements and should not be permitted to divert the work into special fields not contemplated in the basic formulation.

The training in the fundamental branches should be more thorough and carefully proportioned than at present, keeping in mind that the chemical engineer's knowledge must be classified, complete and accurate in the broad principles and not statistical, encyclopediac or of the class of deep profundity required for research in new and original fields. The chemical engineer's knowledge is to be used as a basic "tool" of his trade and may be given as such without detracting one iota from its value as a mental training.

Associated Training.

Associated training is for the purpose of placing in the hands of the chemical engineer a working knowledge of the other engineering branches so that he may utilize them in the development and management of his particular problem. Courses designed primarily as foundational for future advanced study in the same field are not likely to be proper courses for this associated training. Our associated courses should be broad and general, and need not carry the student beyond the foundational principles of the subjects with a working knowledge of the existing design, construction, materials, operation and use of engineering appliances.

The existing methods of teaching by lectures, demonstrations and laboratory work, combined with a liberal use of text and reference books is probably the most effective way of reaching the fundamental and the associated subjects.

Assuming that the student now has a thorough fundamental foundation in the basic principles of his profession and a working knowledge of the associated engineering branches schools should be equipped for *directed* study of their *applications* to the problems of manufacture and production. This applied study is classed as "supplementary training."

In the education of teachers, in the education of doctors, in the study of agriculture, and in the education of all other engineers except chemical engineers, training in the *application* of the respective fundamentals, under the direction and with the assistance of experienced teachers, is considered essential. Medical schools without hospitals, teachers' colleges without practice schools, schools of agriculture without experimental farms, or engineering schools without laboratories and shops would hardly be dignified by classification as schools. Such institutions invariably invest large proportions of their resources in facilities for demonstrating the proper practical applications of the fundamental theories.

Is there any reason why chemical engineering should not and could not be taught by the tried methods of the normal schools, the medical schools, the agricultural schools and other engineering schools? Only with proper chemical engineering laboratories, equipped with real working models of standard appliances to illustrate the basic applications of the industries and a proper curriculum arrangement to permit the student to study and use these appliances and the principles involved in their application will we be on a basis comparable with the methods and the facilities now existing in other educational fields.

But we can accomplish still more with proper laboratory facilities. We can so construct and administer these laboratories that the student will be surrounded by an "atmosphere" of manufacturing and business efficiency, where he will learn by example, by daily contact, by "attrition," the modern methods and practices of office and works management. Chemical engineers should have this knowledge. It is fundamental to the proper fulfilment of their job. Instances where failure has been converted into success and industries saved by a knowlege of proper administrative practices, and the application of business and works efficiency methods, are too numerous to mention. Refinements in chemical processes and ingenious mechanical devices are all ineffective and often useless if inefficient organization and management are permitted to absorb the earnings. Successful chemical manufacturing is based upon three general principles: First, chemical and physical facts; second, mechanical applications; third, organization and management.

The first principle is definite and basic, and is, therefore, not capable of variation, but the second and third are variable and may be far from perfection. The greatest opportunity for increased profit therefore lies in the possibility of mechanical improvements and improvements in the efficiency of the organization. Instruction is provided in a greater or less degree in existing schools in the two first principles mentioned, but the third equally important and fundamental principle is not even undertaken.

The "contact" system which I am urging for adoption as a method for teaching the principles and the applications of chemistry to manufacture, and for imparting a knowledge of the principles and practices of efficiency in organization, management and administration, are the same as the methods which have always obtained, by force of circumstances, in country schools, where the small children in the A, B, C classes unconsciously absorb geography, history, arithmetic, etc., by hearing, seeing and being in the "atmosphere" of the more advanced work. This method can be adopted in the training of chemical engineers and can be made infinitely more effective than in the country school.

The "contact" or "frictional" method would require a laboratory plant larger and more comprehensive than anything yet established, and would have to be managed on somewhat different lines than those now obtaining in engineering schools.

If it is proposed to direct the student in the study of the applications of fundamentals of factories, factory appliances, factory processes and manufacturing business by the "frictional" method, the laboratory should be equipped, organized and administered as a live manufacturing proposition. There should be ground space enough to permit of the erection of one or more complete operating plants in addition to the general laboratories for the study of great applications, such as distillation, evaporation, filtration, wet reactions, high temperature oxidation, electrochemistry, etc.

The complete plants should be chosen from some of the important chemical processes of great common interest, which at the same time illustrate a wide diversity of chemical principles, appliances and products—for example, the manufacture of gas.

These plants should be designed and built from the foundation to the roof so that the structure itself, materials used, equipment and arrangement would illustrate the best known practice and permit of the most economical management and operation. Every structural detail should be provided with a view to educating by example. Office appliances, office sys-

tems and facilities should be provided, and the equipment include books, trade journals, patent literature, samples, advertising matter, house organs, etc., of the business. Conference facilities should be provided in each laboratory for lectures, discussions, demonstrations of the chemical, mechanical and business features of the process. All such work should be conducted in the special laboratories, and the student from the time he enters until he leaves should be made to "stumble over" the tools of his art and become "saturated" with the "atmosphere" of the business from the raw material to the market.

The classes should "go the rounds" of the laboratories in convenient sized sections for conferences and for "manning" the work.

In connection with the chemical engineering laboratory there should be a "shop organization" consisting of competent mechanics, helpers and laborers necessary for the upkeep of the plant, and this organization should be so administered that our students would gain actual experience in dealing with workmen. Office experience, correspondence, dictation, filing and recording, fundamental costing, etc., may all become familiar knowledge to the student by having these facilities arranged for his use and by incorporating them in his daily "business."

Technical Advisers.

These laboratories of engineering chemistry should be organized with a system of technical advisers to be selected from representative experts in each typical field of industrial chemistry. These advisers should be specially qualified experts or active works' managers of live concerns. Their connection with the laboratory should not be merely nominal, but should carry with it a responsibility for the proper equipment of the special laboratory of the industry which they represent. They should also give some lectures upon their specialty, not altogether from the chemical standpoint, which presumably can be covered by the regular instructors, but from the business, organization, and relative industrial standpoint.

The personal contact and acquaintance between adviser and student is an obvious mutual benefit. The attitude of the leading industries toward such a chemical engineering school would become one of personal interest instead of one of criticism.

The technical advisory system, if properly initiated and administered, will ensure modern equipment, methods and instruction in chemical engineering, and will give, through the lectures, first hand and accurate general information on the scope and magnitude of the typical industries. Furthermore, it will definitely dispose of the frequent reference to obsolete and antiquated methods and equipment. By relying upon the judgment of these experts and managers of live and paying industries we would avoid the equally dangerous extreme of becoming loaded up with impractical, untried appliances and uncommercial processes.

Industrial Research.

The largest industrial corporations, such as the General Electric Company, the United Gas Improvement Company, the National Electric Lamp Association, etc., have established and are maintaining research laboratories. These laboratories are directed in their chemical and physical research by able and high-priced men, and are equipped and maintained at great expense.

Such laboratories are being established because there are no existing equipments, on a suitable scale, for solving the problems necessary for their industrial advancement. The corporations have not undertaken the burden and expense of academic research from choice, but from necessity. Our scientific institutions have utterly failed to provide the facilities for industrial research, and as a consequence are being "elbowed" aside from their natural position of technical and scientific leadership.

Under existing conditions the small manufacturer who cannot afford to equip and maintain a research laboratory has no prospect of solving his industrial problems. Unless the scientific institutions develop modern chemical engineering laboratories, equipped with the tools of the industries, and place their facilities within the range of the small manufacturers, industrial development will be limited to the few concerns whose means will permit the establishment and maintenance of laboratories for private research.

On the other hand, with such laboratories as I have outlined, equipped with every facility for experimentation on a large scale, factory managers, technical laboratories and individuals could be interested in availing themselves of these facilities and in establishing researches for solving industrial problems.

Frequent reference has been made to the cost of such a laboratory of engineering chemistry as has been outlined. We are dealing with a great industry and a great problem, and we should therefore exercise our sense of proportion. It is estimated that the minimum net profit of chemical manufactures is $300,000,000 annually. It is also estimated that the combined investment in chemical engineering equipment in all of the educational institutions of the country is not over $30,000, or one hundredth of 1 per cent of the net profit of the industry for one year. A $30,000 investment, or even one hundred times that, is an absurdly small equipment to be devoted exclusively to instruction in the application of chemistry for such an enormous and profitable industry, not to mention the facilities for research development. If the plan outlined will produce men better qualified to enter this field of industry, and men who can "deliver the goods," the question of cost will never prove an obstacle to the development of such laboratories of chemical engineering.*

Columbia University.

North Dakota Lignite has been tested as a fuel for power purposes at the plant of the Wiliston irrigation project of the United States Reclamation Service. The boiler used was of the standard Stirling water-tube type, consisting of three steam drums, one mud drum and three nests of tubes. The conclustions drawn from the test are that steam can be made with a fuel efficiency of 55 per cent to 58 per cent of the heat in the coal, and no difficulty is experienced in getting the full capacity of the boiler. The results compare favorably with those obtained in the average plant using a good grade of bituminous coal, when the heat available to the boiler is considered. Reduction of the moisture in the coal seems to improve economy, but the tests were not sufficient in number to determine definitely the condition of the fuel and time required for weathering to secure the best results. The steam blower for the ashpit is inefficient and there is no gain in supplying superheated steam to it. A saving in steam with equally good results could probably be obtained by substituting a fan. A complete statement of the test is contained in *Bulletin* No. 2 of the Bureau of Mines, Washington, D. C.

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*At the close of this lecture Prof. Whitaker presented a series of concrete questions which will be submitted to all members of the American Institute of Chemical Engineers for a mail vote. These questions are as follows:

(1.) Is the definition of a chemical engineer and the outline of his requirements substantially correct? (2.) Is the prevailing system of education adequate to produce the chemical and engineering talent needed in the industry? (3.) Does the present method of training chemical engineers produce men comparable in their respective qualifications with civil, mining and mechanical engineers? (4) Do factory managers find our graduates qualified to assist in building up and developing the business or are they simply "prospects" who must be "educated" at the expense, in time and money, of the industry employing them? (5.) Are men who receive their experience in the application of the fundamentals in the works where they are employed narrow in their perspective and limited in resourcefulness? (6.) Is there any reason why chemical engineering should not be taught by instruction in the applications of the fundamentals, in a similar manner to other educational systems? (7.) Are the subjects included under "supplementary training" desirable or essential, and is the "frictional" method suggested a proper way to present them? (8.) Would a laboratory of engineering chemistry, equipped and managed as outlined, maintaining a rigid standard for entrance and for graduation, come nearer fulfilling the needs of men in the chemical industries? (9.) Would manufacturers avail themselves of the facilities of such a laboratory for industrial research? (10.) Should such a laboratory be organized and administered, both in its relation to the students and to its work, as a live, up-to-date business?

What Is "Cyanamid"?

By John Woods Beckman.

At the present time there are three different substances that go under the name of "cyanamid." But only one of these has a legitimate right to the name, the two others are illegitimate sisters to this one.

The substance with the right to the name is one whose chemical formula is H_2N-CN, and the name of cyanamid indicates to the chemist that an amid (NH_2) group is combined with a cyanogen (CN) group forming cyan-amid.

Cyanamid is produced in the following manners:

Cyanogen chloride is introduced into a solution of ammonia in ether, and cyanamid is formed according to the following formula:

$$CNCl + 2\,H_2N = CN.NH_2 + NH_4Cl.$$

If tio-urea is treated with the oxides of lead or mercury in an aqueous solution, cyanamid is formed, and the reaction can be written in following manner:

$$NH_2CS.NH_2 = CN.NH_2 + H_2S.$$

Also if a cyanamid salt be treated with an acid under special conditions cyanamid will be produced.

Pure cyanamid is a white, crystalline, delinquescent mass, soluble in water, alcohol and ether. It melts at 40° C. One characteristic feature of cyanamid is its tendency to polymerize, forming di-cyandiamid, and even higher products of polymerization, such as melam. The polymerization of cyanamid takes place when a solution of cyanamid is being concentrated or being kept for any length of time. If cyanamid be heated to 150° C. it polymerizes with great violence.

Chemically cyanamid can be either classed as weak acid or as a weak base. As an acid it will form salts with metals, such as silver, calcium and others. Of these salts the calcium salt is the most important. As a base, cyanamid will form a number of salts; with acids these salts as well as most of the metal salts are very unstable.

Cyanamid, as well as a number of its salts, is an unstable chemical complex that very readily transforms if conditions are right in two different directions, polymerizing forming dicyandiamid, or hydrolizing forming urea.

The substance that is most generally known by the name of "cyanamid" is the product obtained in the process of fixing atmospheric nitrogen with calcium carbide. "Cyanamid" is the name most generally used for this compound, but a trade name of "lime-nitrogen" for this product is being used. The only right "lime-nitrogen" has to the name of "cyanamid" is that one of the components of the product is a calcium salt of cyanamid.

The chemical formula for "lime-nitrogen cyanamid" can only be expressed in the following manner.

$$xCaCN_2 + yCaO + zC + uCaC_2.$$

From this formula it will be readily seen that this compound is a heterogeneous substance and of no definite chemical composition, nor do its components appear in the same ratio all the time.

To explain the composition it is necessary to give, in brief, the chemistry of the formation. If calcium carbide is heated in an atmosphere of nitrogen, the carbide combines with the nitrogen with great avidity, according to the following formula:

$$CaC_2 + 2\,N = CaCN_2 + C.$$

This reaction explains the existence of free carbon in the graphitic modification in the "cyanamid," while the free CaO is an unavoidable admixture. The carbide used for the fixation of the nitrogen contains a considerable amount of free CaO as diluent, and this naturally will appear in the finished "cyanamid." A small amount of carbide is unavoidable, as it is impossible with the present state of the art to totally bind all the carbide within reasonable time.

The chemical nature of "cyanamid" or lime-nitrogen" is a very complex question, and even more so than for the pure cyanamid $NH_2.CN$, on account of the unavoidable CaO. An Italian agricultural chemist by the name of Ulpiani has made some very extensive researches on "lime-nitrogen," and has found that "lime-nitrogen" is permanently changing it chemical structure, even only by storing same and under the influence of ordinary atmospheric conditions. What part the air-slacking of the lime may play in this chemical rearrangement of the chemical structure of "lime-nitrogen," or, more correctly, the calcium salt of cyanamid, is very hard to determine. The calcium cyanamid is of an unstable structure and forms with great ease polymerized as well as hydrolyzed compounds.

In treating the "lime-nitrogen" with water, even with small amounts, still more complex phenomena take place, basic as well as acid calcium cyanamid salts are formed, with the free lime present; urea is formed; calcium cyanamid carbonate may be formed, and a quantity of others, including dicyandiamid and free ammonia.

Ulpiani has made very extensive researches regarding the action of "lime-nitrogen" with various different acids and in various solutions, and these are able to spread a good deal of light over an exceedingly complex chemical problem, which is growing more and more important as new uses for "cyanamid" are developed.

These researches of Ulpiani have shed some light on the fertilizing value of "cyanamid," and he proves quite conclusively that it is not cyanamid or dicyandiamid that is the direct fertilizing ingredient in "lime-nitrogen," but it is the result of hydrolization, the urea, which is acted upon by the bacteria, and ultimately transformed into a nitrogen compound that can be absorbed by the plant.

The third product that claims the name of "cyanamid" bears it only as a trade name. "Lime-nitrogen" is mixed with a certain per cent of sodium nitrate and with a low-grade nitrogeneous material as a diluent to regulate the nitrogen contents of the finished product, and is then called "cyanamid." This material is compounded exclusively for the fertilizer trade and is of no chemical importance.

It is apparent from the above that the name of "cyanamid" is one that has lost its original meaning and has come to indicate altogether different substances to different groups of people.

The name "cyanamid" should stand for the chemical compound known by the following formula, H_2CN_2.

"Lime-nitrogen," although a meaningless name, as meaningless as if gypsum were called "lime-sulphur," may remain the name for the product now generally known as "cyanamid," while for the third product a name like "cyanamid compound" or "lime-nitrogen compound" would probably answer, thus clearing up a question of names that is becoming more and more complex the wider the field stretches in which "lime-nitrogen" can be used.

Niagara Falls, Canada.

Copper-Aluminium Alloy.—A new alloy, believed to consist essentially of copper and aluminium, is being placed on the market under the trade name of cupsor by the Cupsor Company, of 50 Church Street, New York City. It has a color like gold, polishes readily and acquires a brilliant luster and is, therefore, especially adapted for jewelry (watch cases, etc.), and all ornamental purposes. It is further claimed to be highly resistive to chemical influence. Not only is it said to be immune from the attack of the ordinary atmosphere, from salt water and salt air, but it is claimed that sulphuric acid, hydrochloric acid, lemon juice, tomato juice, vinegar, etc., have no effect on cupsor. Cupsor can be readily machined and is easily rolled and drawn into wire. "The tensile strength of cupsor in the casting is estimated to be at least 80,000 lb. to the square inch. The No. 20 gage hard-drawn wire broke in a recent test at 138,550 lb. to the square inch." The specific gravity is 8.11.

Thermal Resistance and Conductance; the Thermal Ohm and Thermal Mho.

BY CARL HERING.

In the preceding issue of this journal (page 676) the writer showed that the degree of temperature and the watt may be conveniently and correctly used to express and measure thermal resistances and conductances, respectively; also thermal resistivities and conductivities. Moreover, they are true mates, as they are numerically the true reciprocals of each other. Hence we have at our disposal two simple, well-known and named units which can be used in such thermal calculations.

The only objection to them is perhaps best shown by the better known electrical analogy. Suppose the ohm had not been named as a separate and independent unit of electrical resistance, and that we therefore were compelled to express and measure resistances in our calculations, in volts, that is, in terms of the number of volts which would be required to cause 1 amp to flow through that resistance. The unit of resistance (equal to the one now called the ohm) would then naturally be 1 volt, hence numerically equal to the unit of e.m.f. This would, of course, be quite correct, and calculations would give the same results as they do now with the ohm, but it would be confusing and more cumbersome; it is mentally disturbing to measure a resistance in terms of volts, because we know that volts represent a different physical quantity than resistances. It unquestionably gives us a greater mental comfort to give this same unit of resistance a name of its own, and one which is different from the unit of voltage, because it is a different physical quantity.

Ohms are equal to the ratio of the volts to the amperes; to measure a resistance in volts is, therefore, making the denominator of this ratio unity so that the numerical value of this ratio is then equal (numerically, but not physically) to the numerator. It is, therefore, a correct expedient to use when we have nothing better. But in electrical calculations it has been found very convenient to create something better, namely, the ohm; it changes nothing, as the unnamed unit is equal to the ohm; it is, therefore, merely adopting a name.

The above analogy applies very perfectly to thermal resistances. It was shown in the article referred to that thermal resistances can be correctly expressed and measured by the drop of temperature for a unit flow of heat (1 watt) just as electrical resistances could be measured by the drop in volts for a unit flow of current (1 amp). But why should we involve in thermal calculations the mentally disturbing features of measuring a thermal resistance in degrees of temperature when we know that physically they are two different quantities? The same is true of watts and conductances. Why not do as was done in electrical mensuration, give these same units names; it will not change anything else; quantitatively they will remain the same.

The purpose of this article is to urge that this be done, namely, that a name be given to the unit of thermal resistance which was shown in the previous article to be the most rational unit, and one which gives the simplest possible relations, namely, unit relations, just as in electrical calculations. Calculations of thermal flows would then be precisely as simple as the electrical ones made with Ohm's law; there would be no reduction coefficient to remember or to use, as it would be unity, like in Ohm's law. It would simplify the conception of a unit of thermal resistance.

To select a new name, like that of Peclet, so as to honor those who taught us the fundamental principles of this branch of science, would be following a very good custom. But the number of such new names is becoming great and requires them to be defined. It, therefore, seems a better expedient to adopt a half-way measure, and name this unit the "thermal ohm." This term was originally sugggested by Dr. A. E. Kennelly.

The thermal ohm, as applied to the quantity above mentioned, is then that thermal resistance which will require a drop of temperature of 1° C. for 1 watt of heat flow.

Ohm's simple law then applies numerically to calculations of thermal flows. If R is the thermal resistance in thermal ohms, W is the heat flow in watts and T the drop in temperature in centigrade degrees, then the thermal Ohm's law is

$$W = \frac{T}{R}$$

which may then be unhesitatingly used in calculations, as the units are then such as will avoid all coefficients and reduction factors.

Specific thermal resistances (that is, resistivities) are then expressed in terms of thermal ohms for 1 in. (or centimeter) cube.

The numbers expressing such resistances in thermal ohms will be the same as the numbers expressing the drop in temperature for a unit of heat current, as explained in the previous article; the only change is in the name; instead of saying that the resistance is so-and-so many degrees, it would be said to be the same number of "thermal ohms."

The specific resistance of copper is about 0.1 thermal ohm for an inch cube; that of graphite about 0.3; carbon about 1.0; iron about 0.5; hot firebrick about 44. The total thermal resistance of the walls of a well constructed furnace of about 100 kw might be of the order of about 0.15 thermal ohm, hence according to the thermal Ohm's law the loss through the walls for a temperature of about 1500° C. would be $W = 1500 \div 0.15 = 10{,}000$ watts. Or conversely, if a 10 per cent loss ($= 10{,}000$ watts) is to be allowed for such a furnace, then as $R = T/W$, the thermal resistance of the walls must be designed to be equal to $1500 \div 10{,}000 = 0.15$ thermal ohm.

According to this same system thermal conductances should be expressed in "thermal mhos" instead of in watts; they will again be numerically the same as the number of watts. Moreover a thermal conductance expressed in thermal mhos would be equal to the reciprocal of the thermal resistance in thermal ohms, just as the electrical mho is the reciprocal of the electrical ohm.

The definition of a thermal mho then is that conductance which will allow 1 watt of heat flow to pass when the difference of temperature is 1° C.

Conductivities will then be expressed in terms of thermal mhos for 1 in. (or centimeter) cube. They will be numerically equal to those heretofore expressed in watts, and are about as follows for a 1-in. cube: Copper, 10 thermal mhos; graphite, 3.3; carbon, 1; iron, 2; hot firebrick, 0.023. The conductance of the above-mentioned furnace will be about 7 thermal mhos.

These units are, therefore, of a very convenient size to use in calculations. The thermal ohm and mho are, of course, independent of whether inches or centimeters are used, just as with the electrical ohm and mho. In stating resistivities and conductivities, however, it is necessary to state whether the numbers of ohms or mhos refer to an inch cube or to a centimeter cube.

Thermal conductivities and conductances are often stated in books and tables in terms of a unit based on the gram calorie per second, the cubic centimeter, and the centigrade degree; this unit is generally called the c.g.s. unit, and is generally the one understood to be used by physicists when they fail to state which one they are referring to. The reciprocal of this is then necessarily the c.g.s. unit of thermal resistances.

To call these the c.g.s. units is, however, an error; they are not really the true and correct c.g.s. units, or absolute units, notwithstanding their common usage as such by physicists. A transmission or flow of heat, whether by conduction, radiation or convection, is energy (work or heat) per second and is, therefore, a physical quantity of exactly the same nature as power, like watts, horse-power, etc., just as the foot and the meter are both measures of the same physical quantity, namely,

a length. But the absolute or c.g.s. unit of power is the erg per second, that is, the dyne-centimeter per second, and not the calorie per second, as assumed by physicists who use the term as above described. This unfortunate error, which is so commonly committed, may, therefore, cause confusion unless an author describes specifically what he means. This confusion is another good reason for abandoning both of them in practical work and using the practical units recommended in this article.

These true c.g.s. or absolute units are, moreover, a very inconvenient size to use in practice. The practical units here suggested are the most convenient to use in all calculations involving electrically produced heat, as in electrical furnaces. The common though incorrect c.g.s. unit—that is, the gram calorie unit—is the most convenient to use in practice when the heat is expressed in calories; it is also the most convenient one for the mathematical physicist under the same circumstances. When, however, the physicist's problem involves electric heat, or, more broadly, a flow, radiation or other transmission of energy of any kind, then the most convenient unit for the mathematical physicist is the true and correct c.g.s. unit described above and based on the erg.

The following tables are a complete set of all the conversion factors and relations between these several units, that is, the new units, the gram calorie units and the true c.g.s. units; the reciprocal values of all these factors are given, so as to avoid all divisions by them; the approximate values are within 2 per cent. They have been carefully checked and cross checked. The unit of thermal conductance referred to in the writer's former papers as "watts" is identical with the one herein called the "thermal mho." And the resistance unit referred to in a previous paper as the "degree" is herein called the "thermal ohm."

Table of Conversion Factors.

(Note.—A flow or radiation of heat may be measured in watts, in gram calories per second, or in ergs per second. In this table the term "c.g.s. unit," that is, the absolute unit, refers to the true ones based on the erg per second; those often called "c.g.s. units" and based on the gram calorie are here given under the latter term. The centigrade degree is understood to be meant in all these units.)

Thermal Resistance.

Thermal ohms:
= centigrade degrees ÷ watts = degrees per watt.
= 1 ÷ thermal mhos.

1 thermal ohm:
= a resistance requiring 1° C. per watt of flow.
= 4.18617 gram calorie units; approx. 50/12.
= 0.0000001 c.g.s. unit of thermal resistance

1 gram calorie unit:
= a resistance requiring 1° C. per flow of a gram calorie per second.
= 0.238882 thermal ohms; approx. 24/100.
= 2.38882×10^{-8} c.g.s. units of thermal resistance.

1 c.g.s. unit of thermal resistance:
= a resistance requiring 1° C. per flow of an erg per second.
= 10,000,000. thermal ohms.
= 41,861,700. gram calorie units.

Thermal Conductance.

hermal mhos:
= watts ÷ centigrade degrees = watts per degree.
= 1 ÷ thermal ohms.

thermal mho:
= a conductance permitting a flow of 1 watt per degree.
= 0.238882 gram calorie unit; approx. 24/100.
= 10,000,000. c.g.s. units of thermal conductance.

1 gram calorie unit:
= a conductance permitting a flow of 1 gram calorie per second per degree.
= 4.18617 thermal mhos; approx. 50/12.
= 41,861,700. c.g.s. units of thermal conductance.

1 c.g.s. unit of thermal conductance:
= a conductance permitting a flow of 1 erg per second per degree.
= 0.0000001 thermal mho.
= 2.38882×10^{-8} gram calorie units.

Thermal Resistivities.

Note.—These are all the reciprocals of the corresponding conductivity units.

1 gram calorie, cubic centimeter, unit:
= 0.3937 gram calorie, cubic inch, unit; approx. 4/10.
= 0.238882 thermal ohm, cubic centimeter, unit; approx 24/100.
= 0.0940478 thermal ohm, cubic inch, unit; approx. 2/21.
= 2.38882×10^{-8} c.g.s. units of resistivity.

1 gram calorie, cubic inch, unit:
= 2.54001 gram calorie, cubic centimeter, units; approx. 10/4.
= 0.606762 thermal ohm, cubic centimeter, unit; approx. 3/5.
= 0.238882 thermal ohm, cubic inch, unit; approx. 24/100.
= 6.06762×10^{-8} c.g.s. units of resistivity.

1 thermal ohm, cubic centimeter, unit:
= 4.18617 gram calorie, cubic centimeter units; approx. 50/12
= 1.64809 gram calorie, cubic inch, units; approx. 5/3.
= 0.393700 thermal ohm, cubic inch, unit; approx. 4/10.
= 10^{-7} c.g.s. unit of resistivity.

1 thermal ohm, cubic inch, unit:
= 10.6329 gram calorie, cubic centimeter, units; approx. 21/2.
= 4.18617 gram calorie, cubis inch, units; approx. 50/12.
= 2.54001 thermal ohm, cubic centimeter, units; approx. 10/4.
= 2.54001×10^{-7} units of resistivity.

1 c.g.s. unit of resistivity:
= 4.18617×10^{7} gram calorie, cubic centimeter, units.
= 1.64809×10^{7} gram calorie, cubic inch, units.
= 10^{7} thermal ohm, cubic centimeter, units.
= 3.93700×10^{6} thermal ohm, cubic inch, units.

Thermal Conductivities.

Note.—These are all the reciprocals of the corresponding resistivity units.

1 c.g.s. unit of conductivity:
= 2.54001×10^{-7} thermal mho, cubic inch, unit.
= 10^{-7} thermal mho, cubic centimeter, unit.
= 6.06762×10^{-8} gram calorie, cubic inch, unit.
= 2.38882×10^{-8} gram calorie, cubic centimeter, unit.

1 thermal mho, cubic inch, unit:
= 3.93700×10^{6} c.g.s. units of conductivity.
= 0.393700 thermal mho, cubic centimeter, unit; approx. 4/10.
= 0.238882 gram calorie, cubic inch, unit; approx. 24/100.
= 0.0940478 gram calorie, cubic centimeter, unit; approx. 2/21

1 thermal mho, cubic centimeter unit:
= 10^{7} c.g.s. units of conductivity.
= 2.54001 thermal mho, cubic inch, units; approx. 10/4.
= 0.606762 gram calorie, cubic inch, unit; approx. 3/5.
= 0.238882 gram calorie, cubic centimeter, unit; approx 24/100.

1 gram calorie, cubic inch, unit:
= 1.64809×10^{7} c.g.s. units of conductivity.
= 4.18617 thermal mho, cubic inch, units; approx. 50/12.
= 1.64809 thermal mho, cubic centimeter, units; approx. 5/3.
= 0.393700 gram calorie, cubic centimeter, unit; approx. 4/10.

1 gram calorie, cubic centimeter, unit:
= 4.18617×10^{7} c.g.s. units of conductivity.
= 10.6329 thermal mho, cubic inch, units; approx. 21/2.
= 4.18617 thermal mho, cubic centimeter, units; approx. 50/12
= 2.54001 gram calories, cubic inch, units; approx. 10/4.

The following table of values will show the convenient size of these two units. It has been calculated from the determinations of S. Wologdine given by A. L. Queneau in this journal, September, 1909, page 383. They are recent determinations of the materials of commerce and do not refer to extra pure or inferior grades. The temperature was about 1000° C. Further details and the method by which they were determined are described in that article. The present writer's opinion is that

some of the heat entering the samples in those tests is likely to have been lost before it was measured. If so, the conductivities given would be too high or the resistivities too low. The figure given above for firebrick was taken from recent careful determinations of Clement and Egy; it gives the conductivity about half as great or the resistivity about twice as great as in the following table; the temperatures were about 400° C. to 800° C. The figures given above for electrode materials were taken from the recent determinations of the writer.

	Resistivities in Thermal Ohms		Conductivities in Thermal Mhos	
	cb. in.	cb. cm.	cb. in.	cb. cm.
Graphite brick	3.76	9.56	0.266	0.105
Carborundum brick	4.06	10.3	0.246	0.097
Magnesia brick	13.2	33.7	0.0755	0.0298
Chromite brick	16.5	42.0	0.0606	0.0239
Firebrick	22.4	56.9	0.0446	0.0176
Checker brick	24.1	61.2	0.0415	0.0163
Gas retort brick	24.7	62.8	0.0404	0.0159
Building brick	26.9	68.2	0.0372	0.0147
Bauxite brick	28.5	72.4	0.0351	0.0138
Glass pot	34.8	88.6	0.0287	0.0113
Terra cotta	40.9	104.0	0.0244	0.0096
Silica brick	46.9	119.5	0.0213	0.0084
Kieselguhr brick	52.3	133.0	0.0191	0.0075

As an illustration of the simplicity of the calculations when these units are used, let the inside of a small furnace be 6 in. cube; let the wall consist of a 4-in. layer of silica brick on the inside and 4 in. of firebrick on the outside. The true average cross-section* of the layer of silica brick perpendicular to the heat flow (namely, the geometric mean, that is, the square root of the product of the extreme sections) for the whole furnace will be 504 sq. in. If its resistivity, from the above table, is 47 thermal ohms, its resistance will be

$$R = \frac{r \times l}{S} = \frac{47 \times 4}{504} = 0.37 \text{ thermal ohm.}$$

A similar calculation for the outside layer of firebrick having a resistivity of 22 thermal ohms gives its resistance as 0.048 thermal ohm. Hence, the total is the sum of these two, which is 0.42 thermal ohm. Incidentally, it shows what a very much smaller insulating effect this much larger outside layer has.

Suppose the drop of temperature through the wall to be 1500° C. Then, according to the thermal Ohm's law, the loss of heat will be

$$W = \frac{T}{R} = \frac{1500}{0.42} = 3580 \text{ watts, or about } 3\tfrac{1}{2} \text{ kw.}$$

This seems high, but the writer fears the resistivities of Wologdine are too low. The heat resistance at the joint, which has not been taken into account, will reduce this loss somewhat. If the arithmetical average cross-section had been used, as is more usual, the calculated loss would have been nearly 50 per cent greater, hence greatly in error.

If it is desired to know the temperature at the joint, then according to Ohm's law the drop of temperature in the inside layer will be

$$T = WR = 3580 \text{ watts} \times 0.37 \text{ thermal ohm} = 1325°.$$

Hence the temperature will be 1325° less than in the inside. As a check the drop in the outside layer will be 3580 watts × 0.048 thermal ohm = 175°, making the total 1325 + 175 = 1500°. This again shows how much smaller the insulating effect of the outside layer is.

To have used conductivities in the above calculations instead of resistivities would have made them quite tedious and much longer.

Philadelphia, Pa.

*"Heat Conductances through Walls of Furnaces." Hering. Trans. Amer. Electrochem. Soc., Vol. 14, 1908, p. 215. This Journal, Vol. VI, p. 495.

The Tooele Smelter.*

By C. H. REPATH AND A. G. MCGREGOR.

The Tooele smelter is situated about four miles east of Tooele City, Tooele County, Utah, and by rail 41 miles from Salt Lake City. The smelter is connected by its own railroad (the Tooele Valley Railroad) with the San Pedro, Los Angeles & Salt Lake Railroad, the junction being about seven miles from the smelter site. The smelter is located upon a hillside, which makes it possible to a large extent for the level of delivery of the product of one building to be the level of charge floor of the next succeeding one.

The Ores.

The ore treated comes principally from the mines of the Utah Consolidated Mining Company in Bingham Canyon. These mines in Bingham Canyon are connected with the smelter by an aerial tramway, four miles long, having a capacity of 125 tons per hour. Custom ores from other mines are received over the San Pedro Railroad and the Tooele Valley Railroad.

FIG. 1.—TOOELE SMELTER. BLAST-FURNACE BINS AT LEFT, SAMPLING AND CRUSHING PLANT IN CENTER, ROASTED ORE BINS AT RIGHT.

Receiving Bins.

The receiving bins are constructed of steel, and are fireproof throughout. The receiving bins have a capacity of 10,000 tons of ore and coal. The bins are so located and arranged that they may be served either by the 50-ton electric larry cars, which deliver the ore from the tramway terminal, or by standard railroad cars.

Sampling Mill and Crushing Plant.

This building is of steel and concrete throughout. It is 40 ft. x 84 ft. in plan and is five stories high. The mill is composed of two sections and each section has a crushing and sampling department. The ore from the receiving bins is automatically fed upon belt conveyors which convey it to shaking grizzlies that discharge the coarse ore into the crushers. The ore from the crushers is elevated to the top of the building by bucket elevators. In the sampling department the ore is cut four times by Brunton automatic samplers, cutting out one-fifth of the amount each time and discarding four-fifths so that of each ton of ore crushed a sample weighing 3.2 lb. is obtained.

The sampling department of each section of the mill contains besides a 12-in. x 24-in. Blake crusher, which is common to the crushing and sampling department:

One O-A sampler.
One 20-in. x 10-in. crusher.
One 1-A sampler.

*A paper read before the Utah Society of Engineers.

One 48-in. x 12-in. rolls.
One 2-A sampler.
One 26-in. x 15-in. rolls.
One 4-A sampler.

The discard from the sampling machines drops into screens of the crushing department, from which it is conveyed to the roaster bins. The final sample is quartered by a quartering shovel, the resulting sample is then dried on a steam dryer, ground in an Englehardt sample grinder, and again ground in a Braun pulverizer, so as to pass through a 100-mesh sieve.

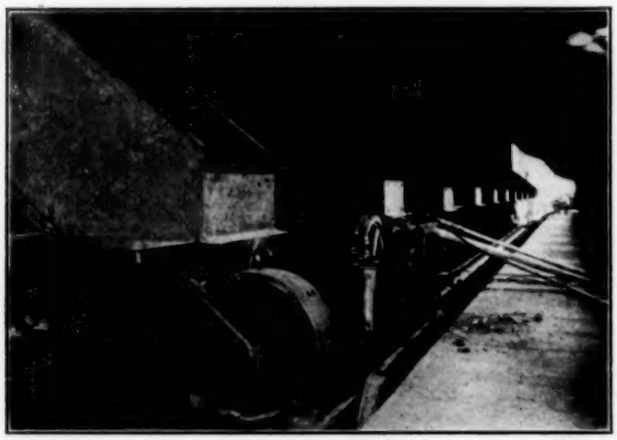


FIG. 2.—CONVEYOR FEEDING MACHINE.

The 100-mesh product is put up in three sample packages, one of which goes to the smelter laboratory for analysis, one to the owner of the ore, and one is filed away for use in case of a dispute.

The crushing department of each section of the mill contains:

One 12-in. x 24-in. Blake crusher.
Two 15-in. x 9-in. crushers.
Two 48-in. x 14-ft. screens.
Two 48-in. x 12-in. rolls.

From the crushing plant the crushed and sampled ore is conveyed by belt conveyors to the McDougal roaster steel receiving bins, which have a capacity of 5700 tons. If it is desired the coarse ore after being sampled may be conveyed to the blast-furnace steel receiving bins, which have a capacity of 3500 tons.

A blast furnace has not yet been installed, but provision has been made for it.

Roaster Plant.

From the roaster ore bins the ore is automatically fed upon a conveyor system which conveys it up to and discharges it directly into the McDougal furnace charge hoppers. The ore in transit from the bins to the roaster plant passes over a Blake-Denison automatic and continuous weighing and recording machine, which accurately weighs all the ore delivered to the roaster plant.

The roaster plant consists of two buildings, each 64 ft. x 162 ft. Each building contains 16 McDougal calcining furnaces of the Evans-Klepetko type. These furnaces have six hearths 16 ft. in diameter and are 18 ft. high. They have revolving water-cooled shafts and arms, driven by a suitable gearing from the bottom. The rabbles are so set as to move the material from the circumference to the center and vice versa on alternate hearths until it finally drops into the calcine hoppers immediately over the tracks of the electric tramming system for transportation to the charge floor of the reverberatory building. No fuel is used other than the sulphur in the crushed ore, the burning of which furnishes sufficient heat to do the calcining. The gases are taken through flues into the large brick and steel dust chamber, which is 120 ft. x 140 ft. x 40 ft. high. This dust chamber is so arranged that the bottom forms a series of hoppers. The entire contents of the dust chamber can be drawn into flue-dust cars of the electric tramming system and conveyed directly to the reverberatory furnaces for smelting. Each of the 32 McDougal furnaces has an approximate capacity of 45 tons in 24 hours.

Reverberatory Plant.

The reverberatory plant consists of three buildings joined together; the reverberatory charge building, containing the ore and coal hoppers over the furnaces, 66 ft. x 280 ft.; the reverberatory furnace building, covering the larger portion of the furnaces, 82 ft. x 280 ft., and the boiler house, 36 ft. x 326 ft. The latter at the present time contains four 746-hp waste-heat boilers and three 350-hp hand-fired boilers. The boilers are all of the Stirling water-tube type. The reverberatory plant contains five coal-fired reverberatory furnaces, the hearth dimensions of which are 19 ft. in width by 102 ft. in length, with a grate area of 7 ft. x 16 ft. These furnaces have a maximum capacity of 300 tons of calcine in 24 hours on natural draft. The fuel used is Diamondville coal, shipped from the mines in Wyoming owned by the Washoe Copper Company. The coal is dumped from large railway coal cars or from cars of the electric tramming system into hoppers that have five points of discharge, directly over the firebox. The flame after leaving the furnace passes through a 746-hp Stirling boiler, which reduces the temperature of the gases going to the main flue to about 600° Fahr. By this means approximately 600 boiler horse-power are obtained from each furnace from the waste heat. The ashes from the furnace firebox fall into hopper cars of the electric tramming system and are hauled away to the ash dump. The slag is skimmed from the reverberatories twice in eight hours. It is allowed to accumulate until its depth is from 3 in. to 4 in. above the skimming plate in the front of the furnaces, and then skimmed into slag cars having a capacity of 225 cu. ft., which are hauled to the slag dump over the electric tramming system. The matte is tapped from the side of the furnaces through a copper tap-hole plate and run through cast-iron launders directly into the converter building.

One of the waste-heat boilers referred to above has been

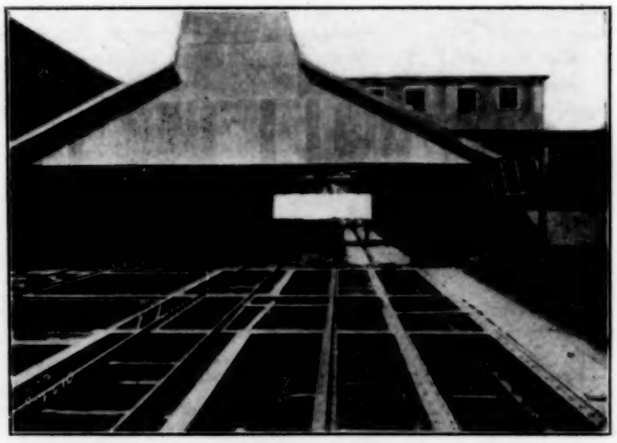


FIG. 3.—REVERBERATORY CHARGE TRACKS.

equipped with a superheater. The superheater is just at the rear of the boiler. In case it proves satisfactory the other waste-heat boilers can each be equipepd with one.

Converter Plant.

The main converter aisle of this plant is 65 ft. x 408 ft., and the casting shed is 52 ft. x 255 ft. The main converter aisle contains five converter stands for 96-in. x 150-in. converters and the converter-lining department. The converters used are of the horizontal barrel type and are 96 in. in diameter by 150 in. in length. The convereters are electrically operated. The main aisle is served by one 60-ton electric traveling crane.

The casting department is served by a 30-ton electric traveling crane. As stated previously, the matte is received in launders directly from the reverberatory furnaces. The end sections of these launders are pivoted so that the matte may be poured

FIG. 4.—SKIMMING FIRST CONVERTER.

directly into the converter opposite a reverberatory furnace, or the launder may be turned so that the matte will fall into a ladle and be transferred to any of the other converters in the building. The slag from the converters is poured into unlined cast-steel ladles and transferred to the reverberatory furnaces by means of overhead cranes. There are two of these cranes, of 12½-ton capacity each. The blister copper is poured into a ladle and transferred by the crane and transfer car to the casting department. In the casting department the ladle is handled by a crane and its contents emptied into anode molds, which completes the operation of producing pig copper. The pig copper contains 99 per cent copper and the gold and silver. It is shipped East, where it is further refined and the gold and silver extracted.

Converter Lining.

The converters are lined in the main converter building, but the lining material is prepared in a building, 24 ft. x 40 ft., adjacent to the ore-crushing plant and receiving bins. The lining material contains a high percentage of silica, having gold and silver values. The material is crushed by the crushers in the crushing plant, and conveyed to a 7-ft. dry pan and pug mill in the clay and silica-mixing and grinding plant. The product of the pug mill drops directly into a bin over the electric tramming system, from which it is transported in cars to the converter-lining plant. The lining plant at the converter contains a 7-ft. grinding and mixing pan. From the grinding pan the material is transferred to the converter, where it is tamped around a cast-iron form by a special Ingersoll-Sergeant tamping machine, 5 in. diameter by 20 in. stroke. The tamping machine is supported by a revolving jib crane, the vertical motion of which is controlled by an electric hoist. After lining the converter is removed to its stand, where it is dried.

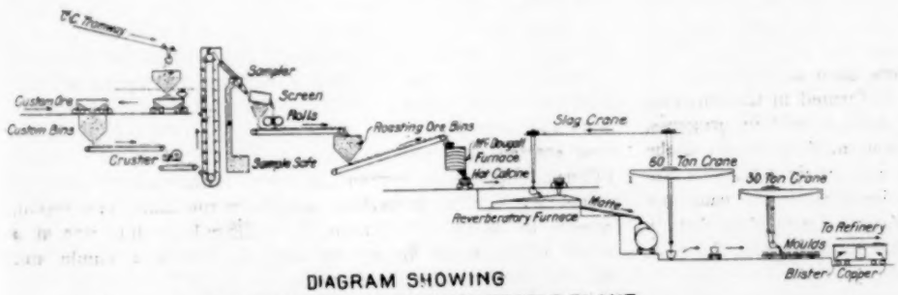


DIAGRAM SHOWING
FLOW OF MATERIAL THROUGH TOOELE PLANT

Smelter Power Plant.

The smelter power-plant building is of brick and steel, the main aisle of which is 52 ft. x 240 ft. It has a lean-to 30 ft. x 90 ft. for the steam auxiliaries, and one on the other side, 30 ft. x 60 ft., for the switchboard and transformer. The building contains the various power engines, blowing engines, compressors and auxiliaries and is equipped with an overhead electric traveling crane.

The building contains:

One 16-in. x 32-in. x 36-in. Corliss engine direct-connected to a 250-kw, 500-volt, direct-current generator.

One 15-in. x 30-in. x 36-in. Corliss engine direct-connected to a 250-kw, 500-volt, direct-current generator.

Two 18-in. x 26-in. x 40-in. x 24-in. vertical, triple-expansion engines, each direct-connected to one 750-kva, 2200-volt generator.

One 15-in. 30-in. x 36-in. x 42-in. blowing engine.

One 26-in. x 52-in. x 52-in. x 48-in. blowing engine.

One 13½-in. x 26-in. x 26-in. x 15-in. x 36-in. steam-driven air compressor.

One 26¼-in. x 15¼-in. x 18-in. motor-driven air compressor.

One 200-kw motor-driven direct-current generator.

One 50-kw motor-driven exciter generator.

One engine-driven 50-kw exciter generator.

Two feed pumps having a total capacity of 678 gal. per minute.

One fire pump of 750 gal. per minute.

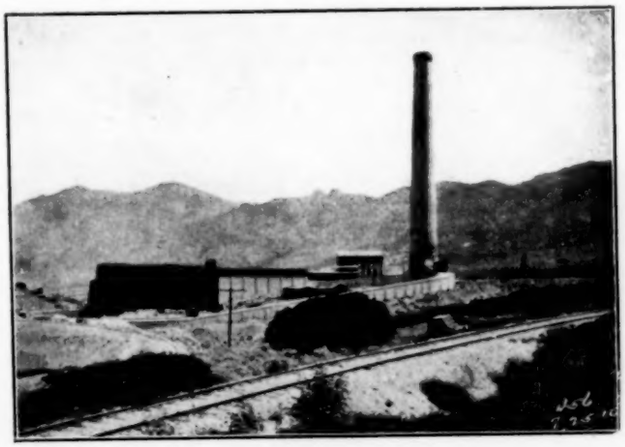


FIG. 5.—FLUES AND STACK.

Besides the above the plant contains the necessary condensers, vacuum pumps, feed-water heaters, traps, etc. The steam for these engines is furnished by the waste-heat boilers and hand-fired boilers adjacent to the reverberatory building.

The engine-driven generators, in addition to furnishing power for the various motors about the plant, will also furnish power for driving air compressors and for hoisting and pumping purposes at the mines of the Utah Consolidated Mining Company.

A cooling tower of the natural-draft type has been constructed for cooling the water from the power-plant condensers and for cooling the water for the McDougal roasting furnaces. The tower is 49 ft. high, and at the base is 20 ft. wide by 140 ft. in length.

Electric Tramming System.

The equipment of the electric tramming system consists of three 7½-ton and two 18-ton locomotives and 50 cars.

Flues and Stack.

The main flue which conveys the gases from the reverberatory furnaces and converters is 20 ft. wide x 18 ft. high x 1360 ft. long. One section of the converter flue is 8 ft. wide x 12 ft. deep and 248 ft. long; the other section has a cross-sectional area of 132 ft. and is 181 ft. long. Both sections are provided with hopper bottoms and chutes for removing the flue dust. The roasting-plant flue is 16 ft. x 16 ft. and is 255 ft. in length. The main and roasting-plant flues are rectangular in section and of brick and I-beam construction, while the converter flues are constructed entirely of steel.

The stack is 25 ft. inside diameter at the top by 350 ft. in height above the base.

The water for various purposes in the smelting plant is obtained from Pine Canyon. The water is conveyed by gravity from a dam in the canyon through a 12-in. pipe approximately 5000 ft. to a standpipe of 50,000-gal. capacity at the plant. The water is distributed from the standpipe to the various departments.

The Status of Metallurgy in Colorado in 1910.

Editorial Correspondence.

Progress in metallurgy in Colorado during 1910 was not marked by any notable innovations, although it is probably true that foundations were laid on which a prosperous industry can be raised in the next few years. The principal subject attracting attention late in the year was the possibility of reviving the mining industry in the State, from which it may be properly inferred that mining and its allied industry metallurgy have been declining for several years past and reached so low an ebb during 1910 as to attract the serious attention of those most concerned.

The decline and low ebb have been the result of a combination of causes, no one of which can be assigned as the sole cause of the condition. The decrease in quantity and quality of smelting ore, coupled with the failure of mine owners to anticipate this outcome and seek new and cheaper methods of treatment ere it arrived, may be given as the principal causes contributing to the present condition of the industry in Colorado. That there are still great possibilities in mining, however, and that metallurgy will play a large part in the success of the future may be gathered from the following presentation of the situation.

Smelting.

Smelting was the pioneer metallurgical process used in Colorado and constituted a thriving industry until the last decade, during which time it has steadily declined. In 1910 the American Smelting & Refining Company had in operation four Colorado plants: Globe (Denver), Pueblo, Arkansas Valley (Leadville) and Durango. These were operated at about half capacity throughout the year. Lead ore was largely brought in from the Cœur d'Alene district, Idaho, while quite a large tonnage of silicious silver ore was shipped into the State from Cobalt, Canada. The smelting of the latter has given rise to some interesting problems, such as the study and treatment of the cobalt speiss which is formed in the smelting process. It is understood that this work is still in progress, and no information is available. A new addition to the Globe plant was completed last year, viz., an arsenic plant for the treatment of flue dust. The process employed is the usual one of roasting and condensation in flues. Local flue dust is treated, as well as some flue dust that is shipped to the plant from Utah.

A number of independent smelting plants have operated during 1910, some of them leading the usual precarious existence of such institutions, and others apparently prospering as well as the status of the industry will permit. Unfortunately for the industry itself, some of these ventures are ill advised, being the creations of promoters who lack information on the basic principles of the smelting business and proceed along lines which almost necessarily lead to failure. It has been observed also, in some instances, that the greater the hazard the less competent the management and the more sudden the termination of operations. Only by the employment of the most skilful technical talent, combined with the most painstaking preliminary investigation, can such misfortunes be avoided.

The Argo plant of the Boston & Colorado Smelting Company, near Denver, was permanently closed in March, 1910. The company had enjoyed a successful career of 40 years' duration, during a part of which time it operated a secret process of refining which made it independent of the larger corporations which sprang up with the introduction of lead smelting. Argo was operated on a copper basis, and owing to the lack of suitable copper ores in Colorado was forced to secure supplies from other States, often at great cost and little profit. The plant has now been completely dismantled by wreckers, who took charge after a thorough clean-up.

The plant of the Modern Smelting & Refining Company, near Denver, which was blown in during 1909, was closed in May, 1910, after less than a year of operation. During this time the plant was occasionally closed for various reasons and finally abandoned as an unprofitable venture. There have been occasional rumors of a revival of activity, but nothing definite has transpired.

In April last, the smelting plant at Golden, which had been purchased and remodeled by the North American Smelter & Mines Company, was placed in commission. With the exception of a few interruptions this plant has been in steady operation, and the prospects are good for continued success. The company has pursued a sound policy, securing excellent technical and business management, and has endeavored to assure a suitable supply of ore by purchasing good mines. The latest acquisition of the company is the Santiago group of mines in the Clear Creek district. As there are apparently ample funds behind the management it is probable that the plant will be operated for an indefinite period. The matte produced is sold to the Globe plant of the A. S. & R. Company.

Still another matte smelter was put in operation at Alma about the middle of the year. Some of the details of this plant were given in our issue for November, 1910, page 610. It is probable that operations will be irregular this winter owing to the difficulty of getting suitable ore supply.

The Salida smelter of the Ohio & Colorado Smelting & Refining Company has been in operation as usual, although at reduced capacity. This plant is the only custom lead smelter in the State which is not controlled by the A. S. & R. Company. It has the usual standard equipment with the addition of the Dwight-Lloyd sintering machines for roasting and agglomerating fine ore and concentrate. The other smelters use the Huntington-Heberlein system of roasting.

An innovation in smelting furnaces was introduced in Colorado last year, known as the Partridge smelter. The change from the standard type of furnace consists in the use of a low-pressure blast of large volume, preheated by the slag flowing from the furnace. An annular opening between crucible and shaft is substituted for tuyères, and the blast is heated in a chamber into which the slag is discharged. A complete description of the furnace and its operation will be found in our issue for December, 1910, page 671. One of these plants has been erected at Como to treat the ore of a mining company hitherto shipping to custom smelters. Other plants are projected for similiar properties, notably in the San Juan region, where, by reason of isolation, it is difficult to ship ores at a profit which might be concentrated by fire in a simple and efficient furnace like the Partridge.

Zinc smelting in Colorado is in about the same status as in 1909. The plant of the United States Zinc Company at Pueblo has been in operation, with six furnaces and 1440 retorts. The manufacture of zinc dust at this plant was one of the important features of the year. The production of zinc in Colorado has increased greatly during the past year, so that the outlook for this industry is excellent.

On the whole the outlook for the general smelting industry is not particularly attractive, for reasons already given, but it is equally certain that other branches of metallurgy will take the place of smelting and tend to expand the business of ore treatment. The decrease in production of ore suitable for smelting in Colorado is shown by the following figures compiled by Franklin Guiterman, manager for the A. S. & R. Company in Colorado:

	1900.	1909.	Decline per cent.
Gold oz.	1,400,000	1,061,000	24.21
Silver oz.	20,300,000	8,900,000	56.16
Lead lb.	82,137,000	32,360,000	60.60
Copper	No change.		
Zinc	Increasing.		
Value	$50,314,000	$30,196,000	38.55

Chlorination.

The mills of the Portland Gold Mining Company and United States Reduction & Refining Company, both at Colorado City, have continued to use the chlorination process on roasted Cripple Creek ores, following this with cyanidation of the tailings. At the former the chlorine is generated from bleaching powder and sulphuric acid, and at the latter by an electrolytic plant of 100 McDonald cells. The barrel charge at the Portland consists of 20,000 lb. of ore, 300 of acid and 100 of bleach (30 per cent available chlorine). At the other plant the ore charge is the same, with the addition of chlorine solution of suitable strength.

It was expected and earnestly hoped by Colorado metallurgists that the dry chlorination plant of the Western Metals Company, at Georgetown, would be tested during the year. Owing to delays incident to the promotion of a new process the plant has not yet been completed and the initial test is delayed. The process to be used has been described in the technical press,* and is usually referred to as the Malm process. Mr. John L. Malm has made what are expected to be vital improvements on the work of earlier investigators, Swinburne, Ashcroft, Baker and Burwell, whereby the process can be profitably applied to base sulphide ores. The plant at Georgetown will afford an excellent opportunity to test the efficiency of the process, as the Clear Creek district has an abundance of such material.

Cyaniding.

There was an increase in cyaniding in this State during 1910, principally on Cripple Creek ores. It has been predicted from time to time that cyaniding ultimately would supplant chlorination, but as noted above it has not yet fully done so. It is true that chlorination has steadily lost favor with Colorado metallurgists, and it is probable that but for the disinclination to abandon a working plant which represents a considerable investment the present chlorination plants would not be in operation and would be superseded by cyanide works.

Probably the most important event in cyaniding last year was the completion and operation of the Cripple Creek mill of the Portland company. The full details of the process have been kept secret, but it is a direct cyanidation of raw ore, with the addition of a substance or mixture, probably a halogen cyanide and an oxidizing substance. The process was worked out by the metallurgical staff of the company. After coarse crushing the ore is ground in cyanide solution in Chilean mills and concentrated on Wilfley tables. The concentrate is shipped to the old mill at Colorado City. The tailing is classified, the sand being rejected and the slime concentrated on Card tables. The Card concentrate is shipped to the Colorado City mill, and the tailing thickened, agitated and filtered. Zinc dust precipitation according to Merrill's process is used. The plant has a number of new devices, such as the Akins classifiers and Rothwell & Akins agitating tanks, and Portland continuous revolving slime filters. The mill is in steady operation on low-grade mine ore, and its successful operation will point the way to the erection of other local mills.

Stratton's Independence mill, which was started in 1909, was in operation last year at about 50 per cent increased capacity. This mill is treating dump ore of an average value of $3.60 at a cost of about $1.50 per ton. The ore is crushed and ground in cyanide solution, then concentrated to remove the sulphotellurides, after which the tailing is cyanided.

The Colorado City mills of the Portland, Golden Cycle and United States Companies were in operation throughout the year, although not always at full capacity. The Golden Cycle uses cyanidation exclusively, while the other two use cyanide only on the tailings from the chlorination plants. The Golden Cycle has probably the best roasting plant in the State, comprising a battery of Edwards' roasters, each having a capacity of 125 tons of Cripple Creek ore daily. The roasted ore is ground to 25 mesh in cyanide solution, passed over blankets and then classified in Dorr classifiers. The sand is leached and the slime agitated and filtered in a vacuum system which was newly installed last year. Precipitation is on zinc shavings.

The only other district in the State where cyaniding has been practised regularly is the San Juan, where the Camp Bird at Ouray and the Liberty Bell at Telluride have been using the process as in the past. There has been a tendency on the part of engineers to extend the use of the cyanide process to other ores and concentrates and during the past year some successful experiments were carried on at Breckenridge and in the Clear Creek district. In connection with this work some new machines were investigated, notably the Bonnot pulverizer and the A. Z. agitator, both of which were found well suited to the work required of them.

The Clancy process of cyaniding was disclosed to the public too late in the year to permit of any tests being made on Colorado ores, but it is certain that the process will be applied to some Cripple Creek ores this year, in an endeavor to test its efficiency. A complete statement of the process will be found in our issue for November, 1910, page 623.

Ore Dressing.

There was probably more activity and new business in this branch of metallurgy than in any other in the State during 1910. The greatest progress was made in the concentration of zinc-lead-iron sulphides, and several new plants were erected and operated. The systems used vary with the nature of the ores, but may be broadly classified into two, viz., those in which roasting is or is not used prior to magnetic separation. Again, the systems employing roasting may use that process either before or after the accompanying wet dressing on reciprocating tables. When roasting is employed low intensity magnetic separators are used, while the high intensity separators are used on raw zinc-iron middlings from wet concentration. The low intensity machines attract the strongly magnetic iron in the roasted mixture, while the high intensity machines attract the weakly magnetic zinc from the raw mixture.

The mills of the American Zinc Extration Company, Leadville; Wilson Mining Company, Robinson; Kokomo Metals Company, Kokomo; Eagle Mining & Milling Company, Red Cliff; United States Zinc Company, Pueblo, and Western Chemical Manufacturing Company, Denver, all use a system of roasting, magnetic separation and wet dressing. With the exception of the Kokomo mill they all use the following general scheme: Crushing, grinding, screening, drying, roasting, magnetic separation of the iron and table concentration of the lead and zinc. At the Kokomo mill the wet dressing comes first, producing lead concentrate, tailing and zinc-iron middling, the last being dried, roasted and magnetically separated.

A difference of opinion exists among metallurgists regarding the relative places of the roasting and wet dressing in the general scheme indicated. Those favoring wet dressing

*_West. Chem. & Met._, Vol. 5, No. 3; Vol. 6, No. 5.

first and roasting with magnetic separation last argue that they immediately get rid of the tailing and save a lead concentrate and zinc-iron middling, the last alone requiring roasting and magnetic treatment. They contend that they thereby avoid unduly loading the drying, roasting and magnetic units with an excess of material that does not require that treatment and consequently reduce the plant required. On the other hand it is contended that slime losses in preliminary wet dressing, and the difficulty in drying the middling, are points against that procedure. Further, that table dressing following roasting is somewat improved owing to the effect which roasting sometimes has of widening the range of specific gravity of the various minerals. It is also claimed that if iron carbonate is present its separation from the zinc is better effected by wet dressing after roasting than before. The two processes have not been in operation long enough to compare their efficiencies or to prove the claims of their adherents.

Several types of roasters are in use at these mills, but the preference is in favor of the multiple superimposed hearth type. A Wilfley vertical furnace was installed at the Kokomo mill, and revolving furnaces at the Leadville plant. At the Western Chemical Manufacturing Company, where sulphuric acid is made from the roaster gases, the furnaces are of the Herreshoff type. The new Wilfley roaster, which was described in our issue for November, 1910, page 646, has not been installed at any plant.

Three types of magnetic separators are used in these mills, viz., the Dings, Cleveland-Knowles and Fairchild. The first two are well known, but the last is a recent device of O. H. Fairchild, superintendent at the Wilson mill, where the machines are in use. There are three sets of magnets of increasing strength arranged transversely to the shaking feed plate. The magnetically separated iron is removed by traveling belts which move around the magnets. In its travel on the belt the iron comes into fields of alternate polarity which cause the particles to change their position and thereby release any mechanically entrained mineral which should not be removed.

Wilfley and Card tables are almost universally used in Colorado, and each has its ardent supporters. At the Leadville mill mentioned above there are some Ford tables in use. In this connection it is interesting to note that during the past summer the Mine & Smelter Supply Company, owning the Wilfley patents, presented its *prima facie* case against the Hendrie & Bolthoff Company, controlling the Card table, in a suit for damages for infringement. The case was heard before a special examiner, but was not finished. Similar suits were instituted against the Deister and Dunham tables.

In the Creede and Clear Creek districts lead-zinc concentration is by wet processes only, as there is but little iron in the ore. Jigs of the Harz type, and Card and Wilfley tables are used. The zinc concentrate from these districts is of excellent quality, ranging from 40 per cent to 55 per cent. The lead is also of high grade, running from 60 per cent to 72 per cent.

The plant of the Empire Zinc Company, at Canon City, has been in operation throughout the year. Raw zinc-iron middling or crude ore is treated at this plant, which is equipped with Wetherill magnetic separators of the high-intensity type. It is understood that the company contemplates the erection of a new plant embodying roasting, magnetic separation and wet dressing. The company has maintained an elaborate testing plant for some time, and has been active in experimental work. The testing plant contains an Elmore unit, Sutton dry table, electrostatic separator and Wilfley roaster.

Tungsten concentration in Boulder County was the principal metallurgical industry of that district last year. The chief difficulty experienced in this work is the tendency of the friable tungsten minerals to slime, thereby causing high losses. The perfection of the Monell slimer last year is said to have reduced these losses materially. The machine is a canvas belt vanner with end motion, and side feed and discharge of tailing. The concentrate is recovered at the end as usual. The principal feature of the machine is its unusual length and large flat concentrating surface on which the mineral particles have ample time to settle with the least possible disturbance in the succeeding washing.

Miscellaneous.

The Colorado School of Mines testing plant was put under construction last fall. This plant, when finished, will be a complete testing mill for ore dressing and metallurgical processes. The fact that the School authorities plan to permit the use of the plant by professional engineers materially widens its field of usefulness. The School will thus perform a service to the State, and the students will have opportunity to witness and assist in many commercial tests on a variety of ores.

The vanadium industry in Colorado has not grown as fast as was expected. The Primos Chemical Company has a mill in operation at Newmire producing vanadate of iron by first roasting the ore with an alkali salt, then leaching and filtering and precipitating the vanadium with ferrous sulphate. Considerable experimental work was done at Boulder last year by Mr. Warren F. Bleecker in an endeavor to improve the process used at present. The results have been gratifying, and our readers will have an opportunity to secure the results of Mr. Bleecker's work by consulting his series of articles which began in our issue for December, 1910.

Portland cement is made in Colorado by two companies, the Ideal and the United States, both operating in Fremont County, at Portland and Concrete, respectively. The shale and limestone used are quarried from beds exposed in the valley of the Arkansas River. Shale and limestone are mixed in the proportion of 2200 lb. limestone to 5800 lb. of shale, making a mixture of about 73 per cent $CaCO_3$ at the mill, or 64 per cent CaO in the finished product. Ball and tube mills are used to grind the mixture preliminary to clinkering, which is done in 120-ft. kilns, using powdered fuel. Fuel consumption is about 140 lb. to 400 lb. of cement. A plaster of Paris plant is also operated by the Ideal Company at Portland. The United States plant is the newer and said to be thoroughly modern.

The discovery of zinc carbonate at Leadville will undoubtedly give metallurgists an opportunity to display their ability in devising treatment for the low-grade ore which cannot be shipped directly to the smelters. Although metallurgical works have lowered the limit on these ores to 20 per cent zinc, there is a vast tonnage that will not run higher than 15 per cent, and there will undoubtedly be a demand for a method of treatment.

The necessity for co-operative work in Colorado metallurgy is becoming more and more apparent. Many districts have an abundance of low-grade ore, which can be mined and treated only by a concerted effort on the part of many small owners. The time has arrived when the mine owner must work out his own metallurgical salvation if his operations are to prove profitable, and if his own property does not of itself warrant a metallurgical plant, he may well consider the advisability of co-operating with his neighbors who are similarly situated. The wisdom of this course has been demonstrated and it should be more widely considered.

Silver Assay.—In addition to being influenced by the temperature of cupellation, the accuracy of the silver assay is also affected by the quantity of litharge used in the fusion. It has been shown that an excess of litharge tends to lower the silver recovered, the litharge probably carrying some of the silver into the slag along with impurities which are dissolved. A flux containing 12 parts each of soda, potash and litharge, 2 of borax glass and 1.1 of flour will give good results if about 80 grams of the same are taken for ½ A.T. of ore.

The Cyanamide Process in the Metallurgy of Gold.*

BY JOHN COLLINS CLANCY.

After the wide publicity given to the Clancy process by the Moore Filter Company, I feel it incumbent upon me to present the following paper for consideration and discussion.

This process is more immediately concerned with the treatment of refractory ores, these ores generally having their values associated with compounds in chemical combination, which are not susceptible to the action of solvents until roasting or some other method of oxidation has disassociated the said compounds and liberated the precious metals. Roasting is by all means the cheapest oxidation process so far known; any attempt to oxidize the base constituents by chemical means is obviously of a very expensive nature, therefore, impracticable.

It is known that, in some cases, profitable extraction of the precious metals from base ores may be obtained, depending on the rapidity of the action of the cyanide upon the finely divided gold in preference to its action upon the base constituents; that is to say, before the base metals are attacked the gold has gone into solution. Sometimes by keeping down the alkalinity of the solution to almost neutral point in "protective" alkalinity a good extraction may be obtained, as the absence of an excess of alkali will protect the metallic sulphides from decomposition and allow the cyanide to work directly upon the precious metals. Of course, these results are only capable of accomplishment when the precious metals are not in chemical combination, unless a solvent for the chemical compounds is present with the treatment solution. The Clancy process to be described in a measure includes the embodiment of these principles.

Preliminary Experiments.

As a preamble, a little history relating to my earlier experiments on refractory ores may not be out of place. The ores of Cripple Creek, Col., may be taken as an example of so-called refractory ore. I need not dwell upon the past or present metallurgical treatment of these ores, but will only state in passing that straight cyanidation on the raw undressed ore yields but a trifle over 60 per cent extraction. It will be apparent, therefore, that a process to extract from about 90 per cent to 95 per cent of the values without roasting or concentration, and at a reasonable cost, would be a boon to the district of Cripple Creek, as well as other districts in different parts of the world. To this task I set myself some years ago, and my improvements may be seen from the different specifications, which show the line of thought and gradations from earlier processes.

The object of my first experiments on Cripple Creek ores was to find a cheap alkaline solvent for the telluride of gold which occurs in these ores, the tellurides existing principally in the form of (AuAg) Te_2, the minerals calaverite and sylvanite being approximately represented by this formula. The minerals calaverite and sylvanite are often found alone, but more frequently associated with pyrite; this close association with the pyrite increases the difficulty of treatment by chemical solutions as well as separation by mechanical means.

I discovered that the alkaline hypochlorites, hypobromites, hypoiodites (NaClO) (NaBrO) (NaIO) dissolved tellurium readily and also attacked the tellurium in the gold telluride crystals, while their calcium salts—the analogues of bleaching powder—were not solvents. Sodium hypochlorite being the cheapest, therefore, there was a possibility that, by crushing the ore fine, the tellurium would be dissolved off and allow of gold dissolution by cyanidation. The equation representing this reaction may take place after the following:

$$Te_2 + 4\,NaClO + 2\,H_2O = 2\,H_2TeO_3 + 4\,NaCl.$$

The tellurous acid so formed would dissolve in the excess of alkali as

$$H_2TeO_3 + 2\,NaOH = Na_2TeO_3 + 2\,H_2O.$$

A preliminary treatment with NaClO, followed by a cyanide treatment on ore crushed to 100 mesh, invariably gave an extraction of from about 80 per cent to 85 per cent of the gold contents, and frequently a higher extraction when the gold telluride in the ore was not closely crystallized with the pyrite.

The preliminary treatment, however, had its drawback, inasmuch as the remnant NaClO left in the pulp after treatment destroyed cyanide unless washed free therefrom. This washing caused extra filtering and meant two sets of solution treatments, and for that reason, together with the alleged high cost of producing NaClO, it was not continued at this time.

My next process was the employment of a soluble persulphate, such, for example, as ammonium persulphate, as a preliminary treatment to dissolve the tellurium contained in the ore. This solvent $(NH_4)_2S_2O_8$ was superior to NaClO, owing to the fact that the iron sulphides did not become oxidized to any appreciable extent while the tellurium was undergoing dissolution.

Another advantage was that the remnant persulphate did not act to destroy the cyanide on subsequent treatment; on the contrary, it hastened dissolution of the gold, after the following equation:

$$4\,KCN + 2\,Au + (NH_4)_2S_2O_8 = 2\,KAu(CN)_2 + (NH_4)SO_4 + K_2SO_4.$$

The high cost of the persulphate, together with the necessity of using two sets of solutions, was the reason for not continuing this method.

I next tried if it was possible to dissolve the tellurium and gold simultaneously by using a mixture of cyanide and persulphate. This scheme was found to work satisfactorily on gold telluride crystals, but when applied to the ore the method was prohibitive on account of the large amount of persulphate required, due principally to the presence and formation in the solution of ferrocyanide, which caused the action of the persulphate to be completely nullified.

The thought then occurred: Could some substance be used to prevent the action of the ferrocyanide upon the persulphate? Reflecting on the behavior of the alkaline hypo-halogen compounds toward KCN, I recalled the weak action exhibited by sodium hypoiodite upon KCN in comparison to that exhibited by sodium hypochlorite and sodium hypobromite; the latter compounds almost immediately converted the KCN into KCNO in alkaline solution—in acid the solution formed cyanogen bromide, cyanogen chloride, respectively.

Guided by this process of thought, the idea of using a soluble iodide, together with the persulphate, in the cyanide solution at once suggested itself. I forthwith tried the effect of potassium iodide in conjunction with persulphate and cyanide solution on gold tellurides, with the result that the tellurides underwent complete dissolution. The nature of the reaction at once explained itself: There being no alkali present in any of these tests, other than the alkali occasioned by the hydrolysis of the cyanide solution, it was simply the persulphate liberating the iodine in presence of the cyanide solution to form cyanogen iodide. The following equations may represent the action shown in two stages:

$$2\,KI + K_2S_2O_8 = 2\,K_2SO_4 + I_2,$$

then

$$I_2 + KCN = ICN + KI.$$

From the study of these reactions I conceived the idea of liberating iodine while the ore was in contact with the cyanide solution by the use of a substance yielding nascent oxygen. Any oxidant might be used which has not a too high velocity reaction, and one which does not unduly increase the alkalinity of the solution so as to prevent the formation and existence of cyanogen iodide in the said solution. Persulphate apparently filled these conditions, as it introduced no free alkali while giving off nascent oxygen, but, on the contrary, neutralized alkali owing to its anxiety to part with one of its sulphions (SO_4) as:

$$K_2S_2O_8 + 2\,KOH = 2\,K_2SO_4 + (OH)_2.$$

*A paper read before the New York section of the American Electrochemical Society.

or we may write the equation in the following manner:

$$2\,KSO_4 + 2\,KOH = 2\,K_2SO_4 + H_2O_2.$$

It will be seen from the above description that the solutions would contain free cyanide together with cyanogen iodide and the oxidizing agent (persulphate), the proportion of cyanogen iodide to cyanide depending upon the speed of the reaction and ratio of persulphate to potassium iodide.

To apply this solution to the treatment of ore and maintain the chemical equilibria necessary for producing cyanogen iodide required very skillful manipulation. For example, the ore had to be made alkaline by applying the necessary protective alkali to the cyanide solution; if excess of alkali was added the persulphate would still liberate iodine, but the iodine so liberated immediately formed alkaline hypoiodite, the hypoiodite formed was rapidly destroyed by the sulphides contained in the ore, thus rapidly using up the oxidizing energy of the oxidizing agent.

The amount of alkali to be added depended, firstly, on the acidity of the ore; secondly, on the proportion of persulphate added, the persulphate requiring alkali in proportion to the removal of its SO_4; thirdly, there was influence of time on the rate of progress of the reaction; fourthly, there was the influence of atmospheric oxygen during agitation accelerating the oxidation of sulphides, which developed acidity and required alkali. To control this alkalinity sufficient alkali had to be added, but not enough to destroy the existence of cyanogen iodide; again, the cyanogen iodide formed would, in the presence of alkali and sulphides, form sulphocyanides, representing a direct loss of cyanide.

Nor is this all; when agitation treatment was completed, the solution after leaving the ore preparatory to filtration would drop, i.e., show a greater cyanide consumption than when in contact with the ore. This consumption of cyanide was probably due to the cyanogen iodide formed, together with the remaining* persulphate, acting upon the cyanide solution in absence of the retarding influence of the reducing agents in the ore.

When it was necessary to add alkali to the solution in order to secure the proper precipitation in the extractors, a further consumption of cyanide took place after the equation:

$$CNI + 2\,KOH = KCNO + KI.$$

I discovered that this consumption of cyanide could be completely stopped, and the cyanide which had combined to form cyanogen iodide regenerated, by the simple addition of sodium sulphite (Na_2SO_3) and lime according to the equation:

$$CNI + Na_2SO_3 + H_2O = NaCN + NaI + H_2SO_4,$$

while the lime added neutralized the acid produced. The addition of sodium sulphite acted perfectly at this point, but it did not take care of the decomposition of cyanogen iodide to alkaline cyanate produced by the necessary presence of alkali in the ore-treatment process (a measure, of course, not allowable in the treatment tank).

It is obvious that the above process would require a skilled manipulator to obtain results at a reasonable cost. In a word, the process is not foolproof. However, if the market price of persulphate would drop to a reasonable figure and a cheap salt were used which would undergo hydrolysis, i.e., give hydroxyl ions in solution to the extent of taking up the acidity produced by the oxidation means employed, then the process would require no more attention than the ordinary straight cyanide process.

Ozone as an Oxidizing Agent.

In view of the high cost of persulphate, my attention naturally turned to the use of a cheaper oxidizing agent, and one that would not introduce, as before stated, alkali in the solution. Ozone, therefore, claimed my attention, as previous experiments with ozone gave interesting results. As there were several ozonizing machines on the market at that time, I began to look around for the most economical machine which produced the greatest amount of ozone at the lowest cost. One machine in particular was guaranteed and backed by a reputable, well-known firm to produce from 100 grams to 150 grams of ozone per kw-hour, and figuring the cost of electrical energy at 1 cent per kw-hour, it seemed that I had now struck the right nail on the head as regards a cheap oxidizer.

An ozonizer of the above description was immediately purchased and installed in my laboratory, whereupon I at once set to work to test the action of the ozone on the treatment of ores with cyanide solution containing soluble iodides, using the ozonized air produced by the ozonizer to liberate the iodine from the necessary chemicals while in contact with the ore.

It would take up too much space to give in detail the results of the numerous tests made on different samples of ore. In short, the ozonized air acted as a first-class oxidizing agent and gave excellent results. The only thing against this scheme was that it consumed too much cyanide unless very carefully watched, it being difficult to establish the proper equilibrium necessary not to allow of rapid oxidation to cyanates. The equation representing the action of the ozone upon the iodide was as follows:

$$O_3 + 2\,KI + H_2O = O_2 + I_2 + 2\,KCH,$$

and, of course, the iodine combined with the cyanide to form cyanogen iodide, as already shown in one of the foregoing equations, the excess ozone forming with the cyanide KCNO after the equation

$$O_3 + KCN = KCNO + O_2.$$

The question now was how to prevent this formation of KCNO

Learning from previous tests on ores containing large amounts of reducing agents that the consumption of cyanide was not so great as with ores which did not contain reducing agents, the thought suggested itself to add a reducing agent to the cyanide solution. I knew that potassium sulphocyanide, although in a sense a reducing agent, had no action on straight cyanide treatment, i.e., the atmospheric oxygen was not absorbed by it. Experiments showed that when sulphocyanide was subjected to ozonized air, cyanide was generated according to the equation:

$$KCNS + H_2O + 3\,O_3 = KCN + H_2SO_4.$$

The alkali present, of course, neutralized the acid as produced, and the cyanide so formed was not decomposed as long as the sulphocyanide was present.

This action opened up a field for thought, and in the ordinary course of thought I tried the action of ozonized air on sulphocyanide solution containing an iodide. Upon introducing the ozonized air into the vessel containing the sulphocyanide and the iodide solution free iodide was at once liberated. When this experiment was repeated, using part of the same original solution, but with a small addition of alkali, on again passing the ozonized aid through the mixture, the solution immediately assumed the characteristic iodine coloration.

This solution was exceedingly active on gold leaf and on telluride of gold, the presumption being that the solution contained cyanogen iodide and free cyanide. The formation of cyanogen iodide by this means, I presume, may be represented by the equation:

$$KCNS + KI + 4\,O = ICN + K_2SO_4.$$

Following up this wealth of discovery, as shown by the above equations, I proceeded with my tests, using the sulphocyanide to prevent the direct formation of cyanate, and was rewarded with most satisfactory results, the consumption of cyanide being very much reduced. However, the amount of cyanide consumption which did take place over and above that consumed by the ore as compared with straight cyanide was probably occasioned by the cyanogen iodide formed being oxidized slowly to cyanate in presence of alkali; the speed in this reaction was not great and probably occurred after the following equation:

$$5\,CNI + 6\,KOH = 5\,KCN + KIO_3 + 2\,I_2 + 3\,H_2O.$$

Potassium hypoiodite (KIO) being formed as an intermediate product, the excess of iodine acting upon more alkali forming

hypoiodite would destroy the KCN formed and eventually become oxidized to KCNO.

Here was a field for the expansion of the process on the large scale, but there is almost always a "but." While the production of ozone was apparently cheap enough as regards the cost of electrical energy, the cost of the machine (installation) to produce enough ozone to treat, say, 1000 tons of ore per day was staggering. The machines required the supervision of trained experts, and for a 1000-ton per day plant a battery of about 500 unit ozonizers would have been required, beside various other mechanical objections too numerous to mention.—"Another hope blasted."

The production of ozone by electrical means is yet in its infancy, and it is only within the last few years that ozonizers have been put on the market to give ozonized air of high concentrations. In the very near future it is probable that ozonizers will be built to give large volumes of ozonized air at high concentration with a small installation of ozonizer units. Meanwhile, I had to look around to find a substitute. How was it to be found? I had already exhausted almost the whole list of economical oxidizers known to chemistry.

Electrolysis.

The employment of the electric current claimed my attention as a means of producing the oxidation necessary while the solution was in contact with the ore. Electrical means for the treatment of ore labors under a stigma, inasmuch as there have been unfavorable criticisms of the use of electrolytic processes, no doubt caused by the numerous failures, among which may be mentioned the electrical precipitation of gold requiring the installation of an enormous plate surface, electrode difficulties and the consequent destruction of the cyanide by the use of the current.

One of the most severe criticisms relates to electrolytic processes in which the direct preciptation of the values from the ore pulp is attempted, the objections probably being due to the scouring of the plates by the circulating ore and the consequent loss of finely divided gold amalgam in the ore pulp.

Much more could be said on this subject, but the Clancy process is not concerned with any electrolytic amalgamating scheme or deposition of the gold on plates or depending upon the dynamical theory of the gold being brought against the cathode electrode and there meeting with a nascent hydrogen or the ore being brought against the anode electrode and meeting its nascent oxygen. The process thus differentiates itself into a method of using the electrical energy to disassociate the chemicals employed and by their agency doing the necessary work.

Electrodes.—Obviously the use of electrolytic methods requires proper anodes. Numerous substances to act as anodes have been tried, such, for example, as lead peroxide, carbon, iron, graphite, etc., but all these substances have been more or less inefficient.

Anodes of lead peroxide under the influence of high current densities break down, i.e., disintegrate. Iron when used as an anode is oxidized by nascent oxygen liberated and forms hydrated oxides of iron, and should the cyanide solutions contain halogen salts the corresponding iron halogen compounds are formed. It is needless for me to describe the many difficulties which I have been up against to find a suitable anode. All the common metals, together with different alloys and compounds, were experimented upon to find a substance which would withstand the action of the halogen compound, i.e., some substance that would not combine with either chlorine, bromine or iodine under the action of the current, so as to allow the liberated halogen or the nascent oxygen to do the useful oxidation work in the solution.

Graphite (extruded) stood the electric current fairly well. i. e., makes a good anode while it lasts, but, under the influence of high current density, it disintegrates and cannot be used for any length of time in circulating solutions containing ore in suspension, as it is worn away by the attrition or friction of the ore particles against the said electrodes. Nor is this all; when it is understood that graphite will not stand a much higher density than, say, 10 amp per square foot of surface, it requires no calculation to figure out the amount of electrode surface required on a working scale.

The only metal which would stand up under the electrolysis of cyanide-bearing solutions containing halogen compounds was platinum. But on a commercial scale the employment of platinum could not be entertained for one moment.

After trying out every known electrode which I thought might prove suitable, I found that oxide of iron which had been melted in the electric furnace was the only thing on this side of Jupiter that would stand up under high current densities and not combine with the halogen compound contained in the cyanide solution. The area of anode surface of this material to give the necessary current density is reduced to a fraction of the amount of surface required in the case of graphite.

In general, these melted iron oxide electrodes used as described are particularly suitable for the electrolysis of cyanogen-bearing solutions and present the following advantages over all other anodes, including platinum:

First, they are cheaper, they have an unlimited life in comparison with the other anodes already mentioned. Second, they furnish nascent oxygen free from carbonic acid, which is of great advantage in the electrolysis of cyanide solutions. Third, they do not disintegrate under high current densities and are so hard that they do not wear away by the atrition of the ore pulp in the agitating tank.

Armed with the necessary weapons in the shape of these highly inert refractory electrodes, confidence in my ability based on previous experiments with platinum electrodes, I felt sure that I could do anything in the shape of electrolysis of cyanogen-bearing solutions containing halogen compounds. I could now use high current densities and liberate chlorine, bromine or iodine in the said solutions without worrying about my anode breaking down.

It will be remembered in my earlier experiments that I was confined to the use of iodine or soluble iodide, as no other oxidizer, even ozone, would liberate bromine or chlorine in the alkaline cyanide solution. Taking advantage of these newfound electrodes I experimented with cyanide, sulphocyanide and a soluble bromide, and obtained excellent results on the most refractory ores by keeping the solutions at the neutral point so as to allow of the formation of bromocyanogen while in contact with the ore.

The use of a bromide, as above described, although giving results equivalent to the use of soluble iodide, was more difficult to control owing to the necessity of maintaining the solutions at the neutral point, and required more electrical energy to do the same work as iodide. Again, by keeping the solutions at the neutral point the formation of cyanate was much more rapid, consequently the free cyanide did not coexist very long in presence of the cyanogen bromide, owing to its high speed of reaction. The bromates produced by the oxidation action would accumulate in the solutions and were apparently not reduced to bromides even after passing through the zinc boxes.

In the case of using a soluble iodide any iodates formed were at once reduced by the action of the sulphocyanides. This reaction between sulphocyanide and an iodate is very striking, so much so that if we take a solution of an iodate and add to it a solution of sulphocyanide, on acidifying this solution, immediate liberation of iodine takes place, or, again, if we take a solution of iodate containing sulphocyanide and electrolyze same, iodine is immediately liberated at the anode.

With bromates this is not the case. Bromates are reduced more or less by the nascent hydrogen evolved at the cathode, but this nascent hydrogen has to do other work, in fact, it has no time to look after the complete reduction of bromates, as during electrolysis a thin pellicle of calcium hydrate is produced on the cathode. The formation of this pellicle is occa-

sioned by the presence of calcium salts in the solution. This pellicle scales off, but is immediately reformed, thus more or less preventing the hydrogen gas in its atomic state from coming into contact with electrolyte and will cause a union of the atoms to molecular hydrogen. When the hydrogen has once assumed the form of molecules then its reducing power is at an end. Bromates, therefore, have a chance of partial reduction in the treatment tank.

The advantages possessed by the use of iodine over bromine are the following: iodine is easily liberated from its compounds at a very low voltage; it does not react violently upon the cyanide solution; it is continuously reduced from its higher oxidation state by the sulphocyanides present in the solutions; it can be liberated from its compound in alkaline solutions, which fact prevents the loss of valuable iodine due to volatilization.

The fact that iodine forms cyanogen iodide in a slightly alkaline solution is of extreme importance in the treatment of ores, as by this fact sufficient free cyanide coexists during the formation of cyanogen iodide. The decomposition of cyanogen iodide to cyanate in the presence of alkali takes place slowly, thus allowing ample time for the precious metal's dissolution.

This method of using cyanide sulphocyanide and a halogen compound in conjunction with the electrolysis of the solution gave excellent results on most of the so-called refractory ores. The cyanide could have been regenerated from the KCNS (i.e., the KCNS which had formed due to the action of the cyanogen iodide upon the sulphides during treatment) but for the presence of the halogen compound.

The reaction $KCNS + 3O + 2KOH = KCN + K_2SO_4 + H_2O$ takes place only to a limited extent. The presence of the halogen salt handicapped this reaction, i.e., it finally became more or less a reversible one after the following equations:

$$KCNS + KI + 2KOH + 4O = KIO + K_2SO_4 + KCN + H_2O,$$
$$KIO + KCN = KCNO + KI,$$
$$KIO + KCN + K_2S = KCNS + KI + K_2O.$$

In a word, the formation of KIO oxidizes the KCN to cyanate, and when regenerating the solution in the presence of ore the KIO acts upon the sulphides in the presence of alkali, producing KCNS.

It will be seen that it is difficult to establish the correct ratios of alkali to obtain an equilibrium to allow of the formation of KCNS and prevent the formation of cyanate. How could this formation of cyanate be prevented? How could the cyanogen-bearing solution be regenerated?

Numerous experiments made with all kinds of additions conducted upon scientific principles with a view to reducing the cyanate to cyanide gave most discouraging results. As far as I am aware, there is no record in technical literature in reference to the regeneration of cyanide from solutions of cyanate. Therefore, I had to launch out into a sea of speculation.

My first experiment was to try the action of the electric current upon a chemically pure solution of potassium cyanate. The action of the current upon straight cyanide (as is well known) results practically in the formation of cyanate. I, therefore, tried the effect of continued effect of electrolysis upon a pure solution of KCNO.

It was not my intention to chase any scientific butterflies, but only the question as to what were the products of electrolysis. I forthwith proceeded with my experiment. On examination of the cathode product NH_3 revealed its presence by its action on the olfactories. At the anode I collected CO_2, the solution meanwhile increasing in alkali. These products for a start were by no means encouraging.

However, as a preliminary to an intended systematic research and to see if there was any possibility of some reduction to cyanide, I withdrew a portion of the liquid from the containing vessel and floated some gold leaf upon the surface of the solution and set it aside for observation, meanwhile continuing electrolysis of the remaining solution. After about 10 minutes this solution became very hot; I was about to throw it out as a spoiled experiment, but, thinking there might be some intermolecular change due to the increased temperature—having in mind among other things Wohler's discovery, viz., conversion of ammonium cyanate into urea—I withdrew a second portion of the solution and floated gold leaf upon its surface, placing it alongside the first experiment.

I watched both solutions carefully. No. 1 experiment, which had already stood 15 minutes, showed no signs of dissolving; No. 2 experiment after 10 minutes surprised me; the gold was surely being dissolved; in 30 minutes it was completely dissolved. No. 1 experiment, after standing all night, showed no signs of dissolution.

Thinking that the reason why No. 1 did not dissolve gold leaf was owing to its not having the same amount of current as No. 2, I set to work and made two experiments under precisely the same conditions, with the exception that one was kept cold by immersion of the containing vessel in a freezing mixture, the other one was allowed to become warm as before. After passing through both the same number of amp-hours of electricity I floated gold leaf on both solutions. On 30 minutes' standing the one that was allowed to become hot dissolved gold leaf completely; the one which was kept cold showed no signs of dissolution even after three days' standing. Platinum electrodes were used in both experiments.

What was the cause of the difference between the electrolyzed hot and cold solutions? Was it the formation of urea? A solution of potassium cyanate containing a small portion of urea I at once electrolyzed, adding a small amount of caustic soda to increase the conductivity; by this means the solution kept at about room temperature. On completion of electrolysis gold leaf was floated upon the surface of this solution and allowed to stand to await results. In about 15 minutes the gold leaf underwent complete dissolution.

From this it would appear that the urea theory was probably correct; or was it the cyanate in conjunction with the urea together with the action of electric current producing cyanide, isopurpurate or some other gold-dissolving compound? Testing with silver nitrate showed an apparently considerable quantity of KCN, but nothing commensurate with its dissolving value on gold leaf.

Cyanamide.

Having made the discovery that urea in conjunction with the cyanate and electrolysis of the solution dissolved gold leaf, the next question was, would any similarly constituted organic compound do the same thing? Urea being an amide compound (carbamide), with the speed of lightning came the thought of cyanamide. I thereupon added a solution of cyanamide (using the calcium cyanamide of commerce, which is soluble to the extent of about 58 per cent to 60 per cent in water) to a cyanate solution, electrolyzed same, treated the solution as before with gold leaf, and dissolution of the gold took place in a few minutes.

With a view to studying this peculiar reaction, I tried the action of electrolysis on a solution of calcium cyanamide; without any admixture other than the addition of a little alkali to increase the conductivity, after electrolyzing for 10 minutes at a fairly high current density the solution dissolved gold leaf in about 15 minutes; this solution also gave a substantial titration with silver nitrate showing probably that cyanide had formed.

Again treating the same solution with the addition of potassium thiosulphate and boiling for the purpose of converting any cyanide present into sulphocyanide, on testing this solution with ferric chloride it gave the characteristic blood-red coloration of the iron sulphocyanide—surely ample verification. But to make it more confirmatory I added a solution of ferrous sulphate to convert any cyanide in the solution into ferrocyanide. On addition of ferric chloride it gave on boiling the characteristic prussian blue.

This satisfied me that some cyanide was formed by the electrolysis of calcum cyanamide, but as to the amount produced

from a given quantity of cyanamide I was in the dark, owing to the peculiar behavior of the titrating solution in presence of organic compounds. It seemed from different experiments made that the production of cyanide by electrolysis of cyanamide solution never rose above three-tenths to four-tenths of a pound KCN per ton of solution.

During all these tests I used solutions of analogous strength to working-scale practice. On referring to Ward's "Dictionary of Chemistry," page 314, line 65, Vol. 2, I found the statement in substance, that cyanamide combines with potassium cyanate to form mono-potassium amidodicyanate of the formula $(CN)_2 NH_2 OK$. Reflecting on this reaction I thought probably that, as the cyanide was formed by the electrolysis of the cyanamide, it was oxidized by the current to cyanate, the cyanate then acting upon the unaltered cyanamide to form potassium amidodicyanate, and potassium amidodicyanate when electrolyzed forms a powerful gold solvent, although it does not give any appreciable titration with silver nitrate.

It will be, I trust, appreciated that to go into the discussion of these reactions would not only require an enormous amount of space, but prove too severe a tax upon my time. Suffice it for the present to say that I have a number of notebooks filled with the most interesting data in regard to these reactions, showing how they branch off and become diversified. I, therefore, propose later on to publish the results of the different experiments and give the theory relating to the data of my notebooks.

To go into practical work is more to the point of this article. However, before leaving the theory, which is always the necessary scaffolding for building up an edifice, I may state that when a solution of calcium cyanamide is mixed with a solution of alkaline ferrocyanide and allowed to stand for a few hours, and even after weeks' standing, it becomes an active solvent for gold, even without the aid of electrolysis, and when applied to the treatment of ores amenable to straight cyanidation gives results equivalent to the simple cyanide treatment even in very dilute solutions.

Further this process can be used on ores which have already been treated by the cyanide process; that is to say, on the residues commonly known as dumps. There exists in these dumps a large proportion of prussian blue, which, when treated with alkali, becomes in substance potassium ferro and ferricyanide or soluble prussiates.

This opens up a cheap means of treating ores amenable to the ordinary cyanide process as well as ores—residues—which have already been treated by the cyanide solution. That is to say, ores in which the precious metals have not been wholly extracted, and would not pay for retreatment by the ordinary cyanide process on account of the cost of cyanide.

The following is an example of using this process upon ore amenable to cyanide treatment:

Two thousand pounds of water containing 1 lb. of calcium cyanamide, 1 lb. of alkaline ferrocyanide, 1 lb. of lime. The ore is subjected to this solution in the ordinary way, say, in proportion of two parts of solution to one part of ore, for a period of, say, 8 to 10 hours, or until extraction is complete.

The following is an example of using the process upon ore which has already been treated by the cyanide process and exposed to atmospheric oxygen which occasions the formation of prussian blue and other ferrocyanogen compounds.

This ore when treated with a solution containing 1 lb. of cyanamide, 1 lb. to 5 lb. of lime (the amount of lime, of course, depending upon the acidity of the ore) in 2000 lb. of water gives results equivalent to the use of straight cyanide solution when used in the proportion of two or three parts of solution to one part of ore.

Again, if this mixture is electrolyzed the solution of the gold is extremely rapid. Calcium cyanamide mixed with alkaline sulphocyanide, electrolyzed, immediately becomes cyanide and dissolves gold rapidly. To show the action of another amide compound, for example, guanidine carbonate mixed with potassium cyanate in conjunction with electrolysis of the solution dissolves gold leaf in less than 10 minutes. I might go on here and cite a whole list of amidogen compounds in conjunction with cyanate solution if space would permit, and I have no doubt this would be interesting to those following this subject.

In the midst of this work on cyanogen-bearing materials the thought occurred to try the combination of calcium cyanamide and iodine. I forthwith made a solution of calcium cyanamide, adding a solution of iodine to see if this combination would dissolve tellurium, and was awarded by finding it to be an excellent solvent. I next tried gold leaf on the same combination and had the satisfaction of seeing the gold undergo dissolution in a short time—a scum of calcium carbonate forms on the surface, preventing more rapid dissolution. From this I deduced that the addition of iodine formed cyanamidogen iodide.

I am unable to find any reference to this compound in chemical literature which I have read, and I, therefore, take the liberty of calling it by the above name and further presume the formula to be CN_2I_2. I next tried the action of cyanogen iodide upon a solution of calcium cyanamide and knowing that cyanogen iodide would be destroyed by an excess of alkali, anticipating that this combination woulld dissolve telluride of gold. Upon applying this solution to tellurium, telluride of gold, these substances at once dissolved, showing that the dissolution was probably due to cyanamidogen iodide, and I again presume the reaction to be in accordance with the formula

$$2\,ICN + CaCN_2 = Ca(CN)_2 + I_2CN_2.$$

Practice of Process.

I will now describe the general method of using the Clancy process on the working scale. The ore is crushed by stamps, rolls, ball mills or any other efficient form of preliminary grinding mills. The degree of fineness necessary for the process being about 100 mesh, this degree of fineness is probably most economically accomplished by the use of tube mills.

The ore is crushed in the cyanide solution containing calcium cyanamide, sulphocyanide and the halogen salts; it is, therefore, under the influence of straight cyanide treatment practically after leaving the rock breakers until it reaches the agitation tank. In the agitation tank it meets with electrolysis, which acts upon the cyanide containing the additional chemicals in solution, forming powerful solvents for the precious metals as already described.

Before describing the treatment solution the manner of dissolving the cyanamide should be known. Calcium cyanamide of commerce comes as a black powder and is about from 58 per cent to 65 per cent soluble in water. It is necessary, therefore, to dissolve the cyanamide in a separate tank and filter from its insoluble residue. A small air-agitating tank is admirably suited for this purpose, which at the same time will act as a cyanamide storage tank. The cyanamide solution may be made as highly saturated as desired and the calculated quantity drawn off when required and added to the cyanogen-bearing solution.

The treatment solution is made up to 2000 lb. of water containing 1 lb. of cyanide, 2 lb. of alkaline sulphocyanide, 2 lb. of calcium cyanamide, ¼ lb. of alkaline iodide, 20 lb. commercial sodium chloride.

I use, after the rock breakers, say, for example, crushing rolls. The product leaving the rolls about 12 mesh is fed into the tube mill and converted into a pulp by feeding the mill with the treatment solution and ore in the proportion of two or three parts of solution to one part of ore.

The discharged pulp after separation of the over size is transferred to the agitation tank to undergo electrical treatment. If it is thought necessary to remove the sulphides before, after or during the treatment the following method presents an ideal scheme:

Let solution containing finely divided ore in suspension be contained in the well-known cone-shaped tank and agitated in

the proportion of two of solution to one of ore, or in the proportion of three of solution to one of ore; if the agitation is then stopped for a few minutes the finely divided sulphide particles settle to the bottom of the cone and by simply opening the discharge valve at the cone apex the sulphide may be completely drawn off together with a small proportion of the non-sulphide pulp. The sulphide product may be run over blankets or similar contrivances and the finely divided concentrate collected. The excess of pulp solution is returned to the agitation tank for treatment.

Here is a means to eliminate the use of concentrating tables and obtain a product of very high value in a very small bulk. Again, the pulp being in a very finely divided state the pyrite or sulphide portion is not accompanied with quartz or gangue —therefore, a clean high-grade concentrate—which result is not capable of accomplishment by the use of concentration tables without the employment of a large quantity of solution with its attendant expense.

The pulp now in the agitating tank carries the correct alkalinity, this being previously established in the tube mill by adding lime so as to contain from 1/10 to 2/10 of a pound "protective" alkalinity per ton of solution.

The conductivity of the pulp is adjusted by adding common salt until the required voltage is obtained. Twenty pounds of salt per ton of solution, as already stated, will invariably decrease the resistance of the pulp so that the voltmeter will register about from 5 volts to 6 volts. In the majority of cases a current of about 50 amp per ton of ore is adequate. It will be easily seen that the cost for electrical energy is not by any means prohibitive.

With iron oxide electrodes it is possible to obtain a current density considerably over 50 amp per square foot of anode surface so that one electrode 3 ft. in length by 3 in. in diameter will be sufficient for the treatment of from 3 tons to 4 tons of ore. In other words, approximately thirty of these iron oxide electrodes would be required for the treatment of 100 tons of ore per day.

If the treatment tank is constructed of iron the tank itself may be used as the cathode. This arrangement would, of course, decrease considerably the cost of installation. The electric generator is the chief item of cost, but a low-voltage generator, say, a 10-volt machine, capable of giving the necessary amperage can be obtained at any of the electrical warehouses.

It is obvious, therefore, that the process may be applied to any existing fine grinding plant provided with agitating tanks. All that is necessary is simply to introduce the electrodes into the circulating ore pulp containing the necessary chemicals and switch on the current. It is essential in every case to maintain the alkalinity, i. e., protective alkalinity at about 1/10 lb. alkali per ton of solution so as to allow of the formation of cyanogen iodide cyanamidogen iodide. About eight hours' treatment under electrolysis usually being sufficient to obtain the necessary extraction.

Regeneration of the Cyanide.—After the eight hours' treatment with the current the pulp solution is brought up to about 1 lb. per ton of protective alkalinity by adding caustic soda and the cyanide contents regenerated up to about 5/10 lb. or 6/10 lb. cyanide per ton of solution. The regeneration of the cyanide is then accomplished simply by giving the pulp two hours more current. It will be understood that the reason for adding the extra alkali is that cyanide regeneration cannot take place in the presence of a halogen compound unless the solution containing sulphides, sulphocyanides and cyanamide is made alkaline.

It will be seen that the whole value of the process depends upon the recovery of the halogen compound. While I have described in the examples a proportion of two parts of solution to one part of ore a proportion of three parts of solution to one part of ore may be used with advantage—that is to say, by using three parts of solution to one part ore a much smaller amount of alkaline halide may be used per ton of solution, thus giving the same ratio of halide salt per ton of ore as in the two of solution to one of ore pulp and consequently a less proportion of soluble halide to be displaced by the water wash in the final slimes cake.

Costs.—In this incomplete description of the process the use of chemicals and current have been dealt with, but no mention of costs has been made. I will, therefore, take the following solution to represent the typical working solution, 2000 lb. of water containing 1 lb. of cyanide, 2 lb. of sulphocyanide, 2 lb. of calcium cyanamide and ¼ lb. alkaline iodide. This appears a formidable mixture when looked at cursorily, but on analysis it does not work out beyond the limits of economic treatment.

For example:

1 lb. of cyanide	18 cents.
2 lb. of cyanamide	6 "
2 lb. of alkaline sulphocyanide	12 "
¼ lb. of alkaline iodide	35 "
Total	71 cents.

This would not represent the total cost of one ton of solution, for, notwithstanding the effect of electrolysis, practically all the haloid salt or salts previously added, together with the sulphocyanide, will be found impaired at the end of the operation, the cyanide and cyanamide alone suffering the necessary decomposition. It is clear, therefore, that no matter what the proportion of solution to ore only the consumption of cyanide and cyanamide per ton of ore is to be taken into account. The amount of cyanide consumption on the ore in presence of cyanamide works out at about 1 lb. of cyanide per ton of ore treated. This consumption of cyanide is regenerated at the expense of 3 cents for cyanamide and at the very outside 3 cents for current (figuring current at 1 cent per kw-hour), making a total cost of 6 cents per ton of ore.

Added to the above cost is the cost of the electrical energy necessary for the electrolysis of the ore pulp. The cost of electrical energy for this purpose works out at about 10 cents per ton of ore treated. This added to the cost for cyanogen-bearing material and regeneration would make a total of 16 cents per ton of ore. These figures would represent the total cost provided that all the solutions were recovered without mechanical loss. From this it is evident that the recovery of the solutions for reuse is a matter of vital importance to this process. The mechanical recovery of the solutions is therefore entirely dependent upon the efficiency of the filter employed.

The Moore filter eminently fulfils the requirements of the process for the recovery of the solution inasmuch as it gives perfect uniformity of cake both in thickness and porosity, which means perfect resistance, and perfect resistance guarantees perfect displacement. In fact, the business combination of the Moore filter with the Clancy process was originally brought about not from any arbitrary or usual business considerations, but because I, after the most rigid examination of all existing filter processes, learned in my demonstrations that this type of filter was the only one that could be depended on to recover practically all of the solutions.

This filter process is too well known to describe it here minutely. Suffice it to say, when the final step in the cycle of treatment is reached—that is, when the cake has been washed with barren solutions—the moisture (containing soluble salts) saturating the cake at this juncture may be completely water washed by giving it the requisite amount of water for displacement. The necessary amount of water for displacement is readily seen without calculation by the diminution of the water level in the wash tank. Further, this may be adjusted to a nicety at the will of the operator, obtaining results not capable of accomplishment by any other known filter. It has been necessary for me to mention the unique points of this filter so as to show how the solutions can be recovered with a minimum of mechanical loss and with a minimum of water

wash. In a properly constructed Moore filter, type A, the loss of solution under proper manipulation should not exceed 10 per cent of the total solution.

It should be understood that the above figures are based on the treatment of rebellious or refractory ores and that they would in all probability be much reduced when dealing with non-rebellious ores.

New York City.

Metallurgy in British Columbia in 1910.

BY E. JACOBS.

The most prominent features in metallurgy in British Columbia are those in connection with the smelting of lead and copper ores. Of nearly a dozen smelteries—one zinc, three lead and the remainder copper—only four are being operated at the present time, namely that of the Consolidated Mining & Smelting Company of Canada, Ltd., at Trail, smelting both lead and copper ores, and the respective copper smelteries of the Granby Consolidated Mining, Smelting & Power Company, Ltd., at Grand Forks; the British Columbia Copper Company, Ltd., at Greenwood (both in the Boundary district); and the Tyee Copper Company, Ltd., on Vancouver Island. There does not appear to be any immediate prospect of other smelting works being operated in the province.

Stamp mills, mostly for gold-quartz ores, have been erected and equipped in various parts of the province, including one with 80 stamps and cyanide plant, at Ymir; one with 46 stamps and cyanide plant at Fairview, Osoyoos mining division; one with 40 stamps and cyanide plant at Hedley; two with 20 stamps each at the Queen mine, Sheep Creek, and the Granite-Poorman mine, respectively, both mines being in the Nelson mining division; one with 15 stamps at the Jewel mine Boundary district; several with 10 stamps each, of which only that at the Second Relief mine, Nelson division, has been worked lately, and some with fewer stamps, chief among these last being that with four stamps at the Nugget mine, Sheep Creek.

Of more than a score of concentrating mills, all but two of which produced nearly $80,000 worth of gold in 1910, which have been equipped for treating silver-lead ores, and several of these for saving zinc as well, less than one-half have been operated during the year, and only two or three the year through.

Stamp Milling and Cyaniding.

The Hedley Gold Mining Company's 40-stamp mill at Hedley, Similkameen, is the only one meriting more than pasing notice at this time. The Ymir and Fairview plants have long been idle; the Granite Poorman, Queen and several smaller mills crush the ore and save most of the gold on the amalgamating tables, and send concentrates to the smeltery to recover that not lost in the tailing. The Jewel mill is provided with a Nichols slimer, but has not yet been run expect experimentally since its recent completion. Passing over all others some attention may be given to the Hedley plant.

Hedley Gold Mining Company's 40-Stamp Mill.—The Hedley mill was erected and for several years operated by the Daly Reduction Company prior to its acquirement last year by the Hedley Gold Mining Company. To date its aggregate of production has been of a value of between $2,500,000 and $2,700,000 from 231,000 tons of ore. A recently published return showed an average assay value of $12.18 for 29,385 tons of ore, and an average recovery of $11.25 per ton, the period covered having been eight months, January-August, of 1910.

The following description has been supplied: The stamps in the four 10-stamp batteries are 1050 lb. each. Formerly they dropped 6 in. at 106 per minute. Last summer changes were made giving a 7¼ in. drop and 99 per minute, one result being a considerable increase in the crushing capacity without requiring the 20-mesh screens to be changed. During recent months new equipment has been put in, the completion of which alters the milling practice to the following: Stamps crush to only 16 mesh, the pulp is classified by eight Bunker Hill screens, the coarse product being concentrated on eight corrugated belt vanners, and the vanner tailing pulverized to 100 mesh in a 100-ton Gates tube mill having Montana-Tonopah lining. The tube mill product unites with the fine from the Bunker Hill screens (running less than 60 mesh), and after being reground passes to classifiers and hydraulic sizers, the coarse from the latter is concentrated on 16 smooth belt vanners, the fine from the sizing passing to 10 Deister tables. The middling from these tables is concentrated over two similar tables. The tailing from the smooth belt machines passes to the sand vats, and that from the Deister tables to the slime vats, this being the method of separating sand from slime. None of the sand reaches the leaching vats coarser than 60 mesh. This new method provides for the single treatment of the sand, the proportion of which is but 25 per cent of the whole. The slime is collected in four 200-ton settlers and pumped thence into six 200-ton agitating tanks in which mechanical agitators are used. The gold solution is decanted from the agitating tanks and treated by two Merrill precipitation presses, in which zinc dust is used. The slime residue in the agitating tanks, remaining after decantation, passes to two Oliver filter presses, the solution to the precipitation presses, and the caked slime to the dump. The first cyanide being introduced into the sand vats and agitating tanks, there is no crushing in cyanide solution. The consumption of cyanide per ton of sand and slime treated amounts to ½ lb. and costs 10 cents per ton. The concentration amounts to 200 tons per month; its base is arsenopyrite, and it runs high in gold. It has been sent to the smeltery at Everett, Wash. The foregoing information is supplied by the superintendent, Gomer P. Jones, and is practically the same as he has presented elsewhere.

Charles Camsell, of the Canadian Geological Survey, in an official report, says: "As the ore is an auriferous arsenopyrite, gold is virtually the only product, and an extraction of about 92 per cent of the total value of the ore is made from amalgamation, cyanidation and smelting. A small percentage of copper, silver and some platinum is known to occur in the ores, and assays show a variable proportion of cobalt, nickel, lead and bismuth. The arsenic is a waste product, so far as the mines are concerned, and no credit is given for it by the smeltery."

About 47,000 tons of ore were milled in 1910, and from this gold to the value of approximately $516,000 was recovered. Power plant and new mill equipment put in during the year cost $115,000.

Silver-Lead and Zinc Concentration.

The Consolidated Mining and Smelting Company's concentrating mill at its St. Eugene mine, Moyie, East Kootenay, is the largest concentrator operating on lead ores in British Columbia. During seven years, 1904-1910, it has treated approximately 1,000,000 tons of St. Eugene ore, from which about 185,000 tons of lead-silver concentrate have been produced, of a total value of $10,250,000. The St. Eugene ore is of a silicious nature, lending itself readily to concentration, which has been about six to one, the concentrate averaging 66 per cent lead. The mill flow sheets show the following: Farrel crusher, picking belt, Blake crushers, rolls, Vezin sampler, elevators, trommels, bull jigs, trommels, rolls, three and four compartment jigs, Huntington mills, Callow screens, sloughing-off tank, Callow tanks, Wilfley tables and Frue vanners. Final middlings from the Wilfleys, containing 20 per cent zinc and 4 per cent lead, are stored for further treatment for zinc. The lead concentrate is shipped to the company smeltery at Trail. The capacity of the mill is 400 to 600 tons daily.

During 1910 an ore-testing plant was practically all put in

It comprises 7 by 9 Blake crusher, set of 15 by 25 rolls, automatic Vezin sampler, three trommel lines, two elevators, four-compartment Richards jig, two Hodge jigs, one each No. 4 and No. 5 Wilfley tables, Wilfley slimer, Overstrom table, Frue vanner, Behrend dry separator, Dings electric magnetic separator, four screens (Callow style), flotation tank, settling tanks, classifiers, etc. A Wetherill separator is being installed, and the putting in of a unit of the Murex magnetic process is contemplated. In addition, the mill is equipped for making cyanide and amalgamation tests. With the exception of these last-mentioned equipments, the mill is designed to make tests on large tonnages of ore, practically all the machinery and plant in it being of the standard sizes used in mills.

A concentrator of 200 tons daily capacity was running at the Blue Bell Mine, Anisworth division, during 1909 and part of 1910. It was designed to treat a large body of ore—more than 300,000 tons—the sampling of which showed an average content of 10.6 per cent lead, 11.7 per cent zinc, and 3.27 oz. silver per ton. This ore was estimated to carry: Quartz and other gangue, 30 per cent; iron sulphides (chiefly pyrrhotite), 35 per cent; zinc blende, 22 per cent; galena, 13 per cent. The products were to be, respectively, lead, zinc and iron concentrates. After leaving crushers and rolls, the ore was elevated to a trommel line, passed thence to two and five compartment jigs and Huntington and Ferraris mills, and the lead pulp thence to Callow screens and tanks and to No. 5 Wilfley tables. The intended separation of the zinc from the iron was not generally successful, so a marketable zinc product did not result. Between 70,000 and 80,000 tons of ore have been milled and more than 6000 tons of concentrate produced, the silver and lead contents of which, however, fell short of the proportions indicated by the above-mentioned sampling.

Of the several mills at Whitewater, Sandon and near Silverton, respectively, that have for several years produced both silver-lead and silver-zinc concentrate, only the last will have notice. In an official report it is thus briefly described: "The Wakefield mill is a well arranged and well ordered plant driven by water power. Jigs operating on 16 mm., 12 mm., and 6 mm. material produce a lead concentrate only, the middlings being recrushed. Jigs operating on 4 mm. material produce a lead and zinc concentrate, the middlings going to a Huntington mill. All smaller sizes are classified, and go to seven Wilfley tables to produce both lead and zinc concentrates. The fiirst Wilfley also makes middlings, which are recrushed. This mill has been operated under lease by the Van Roi Mining Company (Ltd.), of London, England, but its lease having expired, the company is now erecting and equipping its own mill in the same neighborhood.

Concentration of copper ores has been in progress at the Le Roi No. 2 Company's mill, Rossland, and that of the Britannia Mining and Smelting Company on the coast in New Westminster Division. The Le Roi No. 2 mill has a capacity of 80 to 100 tons a day, but usually only half that quantity is put through it. The mill plant includes three Blake crushers, two 6-ft. Chilian (Trent) mills, and four Wilfley tables operated by electric motors. The ore is hand picked; the first-class ore is shipped to the Trail smeltery, while the second-class, constituting about 35 per cent of the whole, is milled. The latter averages 0.4 per cent copper and 0.085 oz. gold per ton, and yields a concentrate containing about an ounce of gold per ton and 2 per cent copper, with an extraction of 75 per cent of the gold and 20 per cent of the copper.

The machinery and plant put in its mill several years ago by the Britannia Company included forty Hartz jigs, Hancock and Wall jigs, Blake crushers, Gates rolls, Huntington, Chilian and Monadnock mills; Overstrom, Wilfley, and Cammett tables (36 in all); 46 Johnston and Frue vanners, Sperry slimers, Richards classifiers, trommels, sand pumps, etc. Some time ago production was maintained at between 400 and 500 tons per day, but latterly the plant has been operated only intermittently, and then chiefly for the purpose of experimenting. In concentrating Britannia ores the result aimed at was the removal of the excess of silica, in order to make a desirable product for smelting with a minimum of copper loss, rather than to concentrate to a high grade of copper. The extremely hard matrix of much of the ore limits the application of concentration. Experience proved that the copper losses in the slimes, resulting from fine crushing, were too great, and that a large part of the ores contained mineral that could easily be separated from its gangue without fine crushing. The practice for some time was treatment which gave a concentrate about neutral as to its iron and silica contents, and from the picking belt a shipping ore carrying a large excess of silica. A modification adopted was to take from the picking belt both the high grade neutral shipping ore (to go to the smeltery crude) and the waste, and to leave the remainder to be crushed for concentration by jigs and on tables.

Smelting Ores Containing Copper.

Granby Company's Smeltery.—The Granby Company's smeltery at Grand Forks is stated to be the largest copper reduction works in the British Empire and among the biggest in the world, second only to the great Washoe works at Anaconda, Mont. It has eight rectangular water-jacketed blast furnaces, six being 44 by 266 in. and two 48 by 260 in. at tuyeres, depth 15 ft. 8 in.; capacity 4000 to 4500 tons of ore per day; 12 Connersville blowers, with a total air capacity of 239,000 cu. ft.; specially designed mechanical charging system; slag haulage by 14-ton steam locomotives and 44-cu. ft. slag cars; three converter stands, 84 by 126 in., operated by electricity, and one stand 72 by 100 in. for which hydraulic power is used, converter capacity being 36,000,000 pounds of copper per annum; three blowing engines, total capacity 19,700 cu. ft. of air per minute, and much accessory plant and machinery.

Conditions at the Granby Company's works are different to those obtaining at other copper smelteries outside the Boundary district of British Columbia in that the Granby Company has to treat an ore giving a very low percentage of sulphur on the charge—as low as 2 per cent. On the other hand, the ore contains silica, iron and lime in such favorable proportions that no barren flux, neither iron nor lime, is required. If it were necessary to use an iron flux in smelting Boundary district copper ores, it would not be practicable to mine and treat them at a profit unless the price of copper was much higher than it usually is, for the iron ore obtainable in the district in considerable quantity does not contain any marketable value. The slag from the Granby furnaces runs from 45 to 48 per cent silica all the time, while occasionally it has contained as high as 52 per cent, which has been the maximum reached. The ore is smelted as it comes from the mine. As a rule, it ranges from 40 to 48 per cent silica. Smelteries in other parts generally have ores running higher in sulphur than they require, but they have the advantage of a low silica content in the ore. It will be seen, therefore, that, as already stated, conditions under which the Granby Company's furnaces are ordinarily aperated are quite different to those attending the reduction of copper ores elsewhere than in the Boundary district.

The low average of Granby ores makes it imperative that labor costs shall be kept down as much as possible, consequently wherever it is practicable to use them, mechanical devices are substituted for the labor of men. This is especially the case in feeding the furnaces, there being no mixing of the charge or other handling of materials between the ore and coke bins and the furnaces. Cars pushed by electric locomotives receive the ore and coke direct from the storage bins and dump them into the furnaces, the mechanical arrangements being such as to admit of this being done quickly and effectively. The company's engineers have devised many ap-

pliances for labor saving, so that economy in working methods is general throughout both mines and smeltery.

It is probable that in the near future changes will be made in the smeltering practice that will admit of a further saving being made in cost of treating Granby ores. The company has lately secured under option of purchase mines that it is believed will yield largely heavy sulphide ores. Heretofore it has been, and still is, the practice at the company's smeltery to produce a 35 to 40 per cent copper matte at first smelting, which involves a loss in the slag of about 5 lb. of copper to the ton of ore. With the sulphide ores above referred to as an aid to the proposed change, it is believed a saving will be effected, and as a consequence more profit be made out of the ores from the company's big mines at Phoenix, B. C. The ore will be smelted to a 10 per cent matte, which will be regraded to a 45 per cent matte for converting. It is thought that not only will the cost of converting to blister copper be less than at present, but by utilizing as fuel the heavy excess of iron and sulphur in the sulphide ores the percentage of coke required in the charge will be reduced. It is expected, too, that the copper loss in the slag will be less by 1½ to 2 lb. per ton of ore, which of itself would mean an appreciably large addition to the profit to be made out of ores from the company's mines at Phoenix. Mr. Wakely A. Williams, superintendent of the smeltery, has been with the Granby Company ten years, during which period he has effected many economies in connection with the handling and treatment of ores received at the works.

British Columbia Copper Company's Smelteries.—During 1910 the British Columbia Copper Company increased the total blast furnace capacity of its smeltery at Greenwood, Boundary district, by one-third, thereby bringing its maximum treatment capacity up to about 2600 tons per diem. This was done by increasing the hearth area of two of the three furnaces by 50 per cent. Previously all three furnaces were 48 by 240 in. at the tuyeres, and their approximate average total capacity had been about 1800 tons daily, with a maximum of nearly 2000 tons. Now they will average between 2400 and 2500 tons per diem.

This increase in smelting capacity has necessitated a corresponding increase in the plant for feeding the furnaces and taking away the molten slag; also an extension of the crane run at one end of the converter building. For slag hauling a larger locomotive, capable of handling two 25-ton slag cars at a time instead of one, as in the past, has been put in. Heretofore there have been in use two 15-ton Baldwin-Westinghouse electric locomotives for slag hauling and three 7½-ton locomotives for furnace charging. An additional charging locomotive has been put in to meet the increased capacity. The slag cars are 25-ton ladle cars, side dumping, and each is equipped with an electric motor having worm gear for tilting the car to dump its contents. The company was the pioneer in using this style of car for slag dumping. The design has since been adopted by the Canadian Copper Company at Copper Cliff, Ontario, and probably a dozen other smelting works, the latest having been the International and the Mason Valley companies in Utah and Nevada.

In the copper converting department of the works at Greenwood there are two converter stands with 84 by 126 2-in. horizontal shells, these taking matte of 45 to 55 per cent copper tenor and producing blister copper 99.3 per cent pure; also containing 20 to 50 oz. silver and 5 to 15 oz. gold per ton. This product is sent to New Jersey for refining. The stands are tilted by hydraulic accumulators, and the shells handled by a 40-ton, four-motor traveling crane. Two more converter shells have recently been added, making a total equipment of seven.

Electric power, transmitted from Bonnington Falls, distant more than eighty miles, is used in the works, in which there are installed motors having a total capacity of about 2000 hp. Among the motors are three of 300 hp, each belted to one of three large Connersville blowers, which supply the blast to the furnaces.

The company obtains the greater part of its ore supply from its own mines and those of the New Dominion Copper Company, which it controls, and it endeavors to maintain an average monthly output as follows: From the Mother Lode, 36,000 tons; Rawhide (New Dominion), 20,000 tons; Wellington Group, 3500 tons; Napoleon at (Boyd's Wash.), 2500 tons. Mother Lode and Rawhide are typical Boundary district copper ores—as they are quite low in marketable metals; Wellington group ore is chiefly oxidized, and having little value other than gold; of the Napoleon ore, the sulphide goes to the smeltery, where it serves as a pyrrhotite flex to reduce the tenor of the copper matte, while the oxidized ore is to be treated in a mill in course of erection near the mine. Provision is being made for increasing the ore supply from the company's mines, and thus render the smeltery independent of custom ores, if these be not obtainable to advantage. The Rawhide output is to be increased to 30,000 tons a month, while 3000 tons will be obtained monthly from the Lone Star, situated just south of the International Boundary line and eight miles from the works. The Lone Star mine is being connected with the Canadian Pacific Railway by an aerial tramway five miles in length; it has an available reserve of 300,000 tons of gold-copper ore of a higher average copper tenor than that from the company's other mines, besides being a good matte maker.

Coke is obtained from southwest Alberta, the railway distance between the colliery and smeltery being 370 miles. With three blast furnaces running, the consumption is 8000 to 9000 tons a month. A reserve of 10,000 to 12,000 tons is kept as an emergency supply in case of interruption in the regular receipt of coke from the colliery.

The approximate figures of production during 1910 are as follows: Material smelted, 456,000 tons. Contents: Gold, 25,640 oz.; silver, 85,900 oz.; copper, 7,351,000 lb. Included in the ore receipts were the following: From the Mother Lode Mine, 352,000 tons; Oro Denoro, 11,473 tons; Wellington Group, 19,200 tons, and Rawhide, 54,400 tons. Mr. J. E. McAllister is general manager, and the number of men employed is 500 to 540, of which total 118 is the normal force at the smeltery.

Tyee Copper Company.—This company's smeltery is situated at Ladysmith, Oyster Harbor, Vancouver Island. Custom ores, chiefly from various parts of the Coast district of British Columbia, southern Yukon, and southern Alaska, have, during recent years, constituted the chief part of the total supply obtained.

At a general meeting of shareholders held recently in London, England, the chairman said in part: "During the year the effects of the low price of copper were to seriously hamper the company's business, mine owners having no inducement to proceed with development, to ship ore freely or to open new mines. Nevertheless the company received 21 per cent more ore than in the previous year. The quantity received, however, had only sufficed to keep the furnaces running 188 days, but as the standing charges were the same whether the furnaces were fully or only partially employed the financial results afforded hope for improvement. Between May 1 and Aug. 1, 1910, the ore received was 19½ per cent in excess of the quantity for the corresponding period of 1909. The manager had reported from British Columbia that negotiations for the acquisition of an important group of properties had been almost closed." Since then it has transpired that the Cornell Mine, one of the Van Anda Group on Texada Island, B. C., has been leased by this company, and that the question of also taking over other property in the group is under consideration. The grade of the ore from the Cornell is shown in the following figures: Production in 1909, 10,178 tons; average assay value, copper (wet), 3.78 per cent; gold, 0.297

oz., and silver, 1.38 oz. per ton. During seven months of 1910 the tonnage was 4501 tons, and the average assay value: Copper, 4.47 per cent; gold, 0.407 oz., and silver, 1.73 oz. per ton. Transportation from the mine to the smeltery is by water, and the trip occupies but a few hours.

Lead Smelting and Refining.

As the Consolidated Mining and Smelting Company, of Canada (Ltd.), smelts both lead and copper ores, and also refines lead, brief notice of all its operations will be put under one subhead. Much of the information here given has been published before, but the present data have been specially revised and corrected by officers of the company.

The company's works are situated at Trail, on the Columbia River, distant from Rossland by railway between ten and eleven miles. Some 550 men are employed. Distinctive features here, in comparison with other reduction works in British Columbia, are the use of (1) the Huntington-Heberlein roasters and converters, and (2) the Betts electrolytic process for refining lead.

An idea of the extent of the company's operations may be obtained from the following figures, which are those for the calendar year 1910: Tons smelted, 445,210. Metal contents: Gold, 138,901 oz.; silver, 2,017,007 oz.; lead, 33,871,837 lbs.; copper, 5,282,139 lbs. The figures for the company's fiscal year ended June 30, 1910, were generally higher, especially in lead produced (42,368,816 lb.) while total value was nearly $400,000 greater.

Lead Smelting.—The lead ore supply is mainly heavy sulphide with comparatively little dry ore, the lead furnaces carrying a charge of from 35 to 45 per cent lead, as compared with the customary 10 to 15 per cent. More than 80 per cent of the ore received runs from 20 to 75 per cent lead, the remainder consisting of dry gold and silver ores, a small quantity of oxidized lead ore, and some pyritic mill concentrates. The lead beds are made up with heavy ores and concentrates only. They contain about 600 tons each, with 38 to 40 per cent lead. The sulphide, with lime rock, lead matte, etc., is treated in the Huntington-Heberlein roasters and converters. The resultant lead sinter is smelted in the blast furnace with dry ores, etc.

There are seven Huntington-Heberlein circular roasting furnaces 26 ft. in diameter, with hearths revolving once in three minutes. Stationary cast-iron rabbles work the ore through in about two hours. The making up of the roasting charge is important. An approximate analysis of the charge as now run is as follows: Lead, 40 to 44 per cent; iron, 10 to 13 per cent; silica, 8 to 11 per cent; lime, 7 to 10 per cent; zinc, kept under 10 per cent, if possible. Pyritic concentrates and matte form part of the charge. The converters number 24. They are 8 ft. 8½ in. in diameter, the diaphragm in four sections, bolted together. The time of converting varies widely with the charge, but averages three 10-ton charges in twenty-four hours, with a 10 to 12 oz. blast, which is decreased to 2 oz. at the close. Properly made up, carefully fired, roasted, and converted, the product is fairly hard, yellowish or grayish yellow, showing some litharge, and is easily broken up with little fine material. In the furnace it runs with cool top and bright tuyeres.

Two lead blast furnaces are, respectively, 45 by 160 in. and 45 by 140 in. at the tuyeres. One of these has smelted as much as 260 tons in a day, but the average is about 190 tons of ore, beside by-products, matte, slag, etc., these constituting 20 to 30 per cent of the charge. The height of the charge is 17 ft. 6 in. above the tuyeres, and the blast used is 32 to 34 oz. The charge usually has about 85 per cent of Huntington-Heberlein roast, varying considerably with the ores, and some oxidized and silicious dry ore and lime rock. The coke is about 12½ per cent of the charge, not taking into account slag and easily melted by-products. An addition of 100 to 300 pounds of slag from the high-grade copper furnace is made to each charge to keep it open, make the furnace action uniform and reduce the percentage of zinc. The lead in the charge averages about 40 per cent, but it has been successfully run as high as 55 per cent. Sulphur is kept below 4 per cent, the lower the better, as the matte fall is less. The speed of running decreases markedly with a small increase of sulphur. From 30 to 40 per cent of the sulphur is eliminated in the blast furnace.

The slag is kept at about 31 to 33 per cent silica, 18 to 20 per cent lime, 24 to 30 per cent iron and manganese, 7 to 12 per cent zinc oxide, and 8 to 16 per cent alumina. It contains about 0.4 oz. silver and a little more than 1 per cent lead. With high zinc the lime is reduced to keep the slag fluid. It is not profitable to use silicious ores to bring the slag much above 33 per cent silica, as the furnace runs too slowly. Alumina gives no serious trouble. Iron and manganese may vary between the percentages above given without much difference in the furnace running. Scrap iron is added occasionally, although unnecessary when the charge is good.

The lead bullion flows from the side of the furnace into a kettle of fifty tons capacity, from which it is pumped into vertical anode molds. The matte and slag are tapped into a large receiver at the end of the furnace, the slag overflows into a granulating launder, while the matte is tapped from the receiver into pots. A single day's production of bullion has been as much as 130 tons, but about 75 tons is the present rate of production, with one furnace running. An average analysis of the bullion is: Copper, 0.22; manganese, nil; zinc, 0.098; antimony, 0.32; arsenic, 0.28; nickel, cobalt, and cadmium, nil; bismuth, 0.0133, and lead, 98.5 per cent; silver, 100 oz., and gold, 0.5 oz. per ton. A furnace is operated continuously for about seven months, then if accretions have accumulated sufficiently to make it profitable to blow it out and clean up the other furnace is blown in.

Lead Refining.—In the electrolytic refinery there are 240 tanks, each 3 by 8 by 3½ ft., arranged in cascades and pumps to keep the electrolyte circulating. Each tank has 20 anodes, which are spaced 4¼ in. apart in the tanks. Cathodes, consisting of sheets of pure lead, about 1-16 in. in thickness, are placed between and outside the anodes, 21 in each tank. The electrolyte used is lead fluosilicate with free fluosilicic acid, averaging 12 per cent acid and 5 to 6 per cent lead. The tanks are operated with a current density of 16 amp. per square foot of cathode area. The voltage averages 0.32 volt per tank, the contacts causing a loss of not more than 0.02 per tank. A tank is worked out in about eight days, and 15 per cent of the anodes, as scrap, goes back to the melting pot after having been washed. The cathodes are removed from the tanks, washed, melted, and cast into pigs of lead of 190 lb. each for the Chinese trade, and half that weight for other markets. An average analysis of 2,000 tons of lead ready for shipment was: Arsenic and bismuth, nil; zinc, 0.0005; silver, 0.0013; copper, 0.00075; lead, 99.9938; iron, 0.00075; tin, 0.0001; antimony, 0.0028.

The slime from the anodes is agitated in tanks with hot water, filtered and dried, and next melted in a water-jacketed reverberatory furnace lined with magnesite brick. The impurities are oxidized off, and doré metal running 960 to 975 parts gold and silver is obtained. The doré metal is parted with sulphuric acid. The silver sulphate is run into steam-heated tanks, where the silver is precipitated as a slime by bars of metallic copper. The silver and gold are collected, melted and cast into bars. The gold is 995 fine and the silver 999, the latter in bars of about 80 lb. each. Fine gold and some silver are sold to the branch of the Royal Mint, Ottawa, Ontario, and the United States Assay Office at Seattle, Wash. Silver finds its chief market in the Orient; refined lead is sold in eastern Canada, the Orient and other places.

By-products are copper sulphate and antimony. The copper sulphate solution, obtained in the course of treating the slimes, is crystallized into copper sulphate, making a commercial product used by farmers to kill smut in seed wheat.

Copper Smelting.—There are five copper blast furnaces, as follows: Two of 42 by 300 in. at the tuyeres; one of 42 by 263 in., and two of 42 by 240 in. The height of the charge is 9 ft. above the tuyeres, and the furnaces are operated with air pressure of 36 oz. Two forehearths are used with each furnace, but only from 0.25 to 0.50 per cent of the total matte is caught in the second settler. Most copper ores are smelted raw to a low grade first matte. The slag from the low-grade furnaces runs: Silica, 43 to 46 per cent; FeO, 14 to 15 per cent; lime, 20 to 22 per cent; copper, 0.1 per cent, and a very little gold and silver. The first matte, containing about 15 per cent copper, 27 per cent sulphur, and 56 per cent iron, is granulated, roasted in O'Hara furnaces to 10 to 12 per cent sulphur, blown in Huntington-Heberlein converters to a product which runs 3 to 4 per cent sulphur, and then smelted in a high-grade furnace, with suitable ores, to a matte running about 40 per cent copper, which last product is shipped to Tacoma, Wash., for bessermerizing. The slag from the high-grade furnace analyzes: Silica, 40 to 42 per cent; FeO, 26 to 28 per cent; lime, 16 to 18 per cent; alumina, 10 to 12 per cent, and copper, 0.35 per cent, with gold 0.02 oz. per ton. Some of this slag is used in the lead furnaces and part of its gold content recovered there.

General.—The ore supply comes chiefly from East Kootenay, Slocan, Nelson, Rossland and Boundary districts, and some from the neighboring State of Washington. Coke, of which about 225 tons is used daily, comes from Hosmer, Crow's Nest Pass, 249 miles distant. Limestone, chiefly for flux, is obtained from the company's quarry at Fife, 68 miles away. Requirements are about 200 tons a day, and cost is $1 per ton in the charge bins. Electric power is transmitted from Bonnington Falls, 33 miles distant, at 20,000 volts. At the smeltery it is stepped down to 550 volts. The total power consumption is about 3000 hp. The works are advantageously situated, with railway connections to all important mining camps on the Kootenay and Boundary districts, and with American roads to the southward. Mr. W. H. Aldridge is managing director of the Consolidated Mining and Smelting Company, of Canada (Ltd.), and has his headquarters at Trail.

New York Section American Electrochemical Society.

A meeting of the New York Section of the American Electrochemical Society was held at the Chemists' Club in New York City on Dec. 16, 1910. The chairman of the section, Mr. E. A. Sperry, presided. The meeting was very well attended.

The first part of the meeting was devoted to a general discussion of the fixation of atmospheric nitrogen, introduced by a brief paper by Dr. E. F. Roeber, who reviewed in succession the principles first of those processes in which air is treated, as it is, by means of electric spark or arc discharges (Bradley & Lovejoy, Birkeland & Eyde. Pauling, Badische Company) and then those processes in which the nitrogen is first separated from the oxygen in the air and afterwards caused to enter into combination with something else (manufacture of calcium cyanamide and synthesis of ammonia). As to the comparative power consumption in the manufacture of nitrates or nitric acid by arc discharges and in the manufacture of calcium cyanamide the following figures were given as representing best commercial practice: One kw-hour binds 17 grams of nitrogen in form of HNO_3 and 70 grams of nitrogen in form of calcium cyanamide. (In the latter case the power consumption in making calcium carbide is included.)

In the quite extended discussion which followed Messrs. Baekeland, Beckman, Loeb, Richards, Sperry, Tucker, Weedon, Whitney and others participated.

Mr. J. C. Clancy then presented a paper on his new process on the substitution of calcium cyanamide and electrolysis for cyanide in gold metallurgy. The paper is printed in full elsewhere in this issue.

Separation of Oxygen by Cold.*

By JAMES SWINBURNE.

This industry has arisen, almost by accident, out of that of the liquefaction of gases. It had long been the ambition of scientific men to liquefy first one gas and then another; the liquefaction of such gases as oxygen, hydrogen and nitrogen being a sort of North Pole to be reached. It is doubtful whether reaching the North Pole is of any value to anybody but newspaper men, whereas the production of very low temperatures was bound to increase our knowledge very materially. Like most researches that appear at first to have purely increase of knowledge value, low-temperature work has proved to be, in addition, of direct commercial importance.

Though the separation of gases by cold has arisen out of the liquefaction of air, the two problems are not quite the same. Leaving out of account the details of the machine employed for the moment, we may consider the apparatus as a closed vessel, insulated to prevent heat getting in through the walls and having three pipes. At one pipe we may assume that air at the temperature of the room, but at a higher pressure, enters the machine; at a second and third, oxygen and nitrogen come out at the pressure and temperature of the room.

Air under high pressure has less internal energy than the same mass of air at the same temperature and lower pressure, so that if air is continually going into the apparatus at the temperature of the room and at high pressure, and its constituents are coming out at the same rate at the temperature and pressure of the room, more energy is coming out of the apparatus than is going in.

This is impossible as a permanent regime. It cannot be put right by less air coming out than goes in, because that would mean an accumulation of air. A balance can be obtained by taking out most of the air at the pressure and temperature of the room, with a high internal energy per kilogram, while a little of the air comes out at a low temperature, with less internal energy per kilogram. This portion may be quite small and may be liquid.

But the air is supplied under pressure and comes out at the pressure of the atmosphere, so it must be expanded inside the apparatus, and as there is no way of getting the energy of expansion out of the apparatus as described, it must be converted into heat, provided it is expanded irreversibly. This means loss of power, which is a small point, and production of heat just where it is not wanted, which is a serious matter.

If, therefore, in addition to the three pipes, there is communication by a shaft, and the apparatus may be arranged so that the expansion of the air works some sort of engine which turns the shaft so that work is given out, the yield of the liquid air must be larger. The ideal liquid-air machine would, therefore, take in compressed air and give out air at the same temperature, but at the pressure of the atmosphere; and a stream of liquid air, and as much power as possible, would be given mechanically through a shaft or other mechanical transmitter. It will be explained presently that the Linde separation process is nearly reversible, capable of being but little improved by giving out energy.

It is generally assumed that the problem of the separation of gases is substantially the same. This is not the case, however, and the overlooking of the difference has led inventors and designers of such apparatus into a series of absurd mistakes.

Take the ideal apparatus for separating gases; we have a pipe leading in gas under pressure at the temperature of the room, and two pipes coming out; one carrying oxygen more or less pure, and the other nitrogen more or less pure, at the temperature and pressure of the room. Now the internal energy of the gases coming out is greater than that of the compressed gas going in, and if the energy were alone to be

*A Faraday Society paper, presented at the December meeting in London.

considered, once the apparatus was started and got into running order as regards temperatures, air might go in at the pressure and temperature of the atmosphere, and come out as oxygen and nitrogen at the temperature and pressure of the atmosphere, the process being continuous and costing nothing appreciable. But as the apparatus cannot be made quite heat-tight in practice, the air might have to go in under slight compression, so as to have slightly less internal energy than the gases coming out; the difference being just enough to compensate for the leakage of heat through the casing of the apparatus. This is an ideal apparatus which several inventors have designed; but it is not possible to make such an arrangement work.

The energy is not the only quantity to be considered. Let us return to the simple case of an apparatus which is perfectly heat-insulated, fed with air at the pressure and temperature of the room, giving out oxygen and nitrogen separately at the same temperature and pressure. The entropy of the separated gaess is less than the entropy of the air going in, so if the process is continuous there must be a continual disappearance of entropy inside the apparatus, which is not thinkable. Air at a higher pressure, but at the temperature of the room, has less entropy, so it may be supposed we can get things right by compressing air until it has the same entropy as a compressed mixture, as the issuing gases have as they come out. We would then have an entropy balance.

This will not suffice in the ordinary course, however, because the compressed air must be expanded, and if that is done irreversibly inside the apparatus, more entropy is produced there, and to make a proper entropy balance either the expansion must be done reversibly or air must be so compressed that its entropy is reduced below the sum of that developed by any expansion and that of the separate gases. Imagine this done; have we a thinkable machine? No, because now our energy balance is wrong; we have air going in with less internal energy than the gases coming out.

It will be found on further consideration that the energy balance is got right by using imperfect heat insulation, heat leaking in to make up the extra energy of the issuing gases over that of the entering air; the entropy of the compressed air plus the entropy due to the heat leaking in, plus any entropy due to irreversible expansion or conduction of heat in the interchanger, being equal to the entropy of the gases coming out. If desirable the case may insulate perfectly, and some of the air or gases may be delivered continuously in the liquid state. The point to notice is that as the oxygen and nitrogen coming out have less entropy than if they were mixed at the same temperature and pressure, the apparatus has to be fed with compressed air, and energy is therefore needed to separate the gases. This point has been so often overlooked that it is necessary to dwell upon it. As far as I know, I was the first to publish it.

It has sometimes been supposed that because the gases are separated at a very low temperature the data determining the energy necessary to separate them do not depend on the difference of entropy of the separate gases and the mixture at the temperature of the room, but at the temperature of separation. This is erroneous and may give rise to further mistakes. That it is erroneous is clear from the illustration. Suppose the difference of entropy of cold gases separate and cold gases mixed is less than the corresponding difference at the temperature of the air; even if the entropy of the entering air is reduced enough to compensate for this only, and not for the entropy of the issuing gases, the entropy balance of the machine as a whole is wrong.

Heat leaking in is difficult to value, as there is an irreversible change as the heat is conducted to the cold gases, which means increase of entropy; and there is increase on entropy, due to irreversible expansion. For simplicity take a heat-tight case, and the energy balance kept right by drawing out a small stream of liquid air. Then, if there is no irreversible process inside the apparatus, there is no growth of entropy in it; and the entropy of the gases coming out, plus the entropy of the liquid coming out, must be equal to the entropy of the air going in. In other words, the entropy balance depends on the entropy of the issuing and entering streams, and not on the entropy of the gases at some inside point where they are at very low temperatures, though the separation actually takes place there.

To make the matter clearer it may be better to put the entropy question more fully. The change of entropy of a gas may be split up into entropy of change of volume, which varies as the logarithm of the volume or of the free path of the particles, and entropy of change of temperature, which varies as the logarithm of the speed of the particles. We may call these two the volume and the speed entropies ϕ_v and ϕ_s. When two gases mix, the volume of each increases in a way—that is to say, it pervades a larger space, and the molecular free path of each particle is increased if you take into account the collisions with other particles of the same gas only. The mixture of two mutually inert gases thus involves increase of entropy, which I will call entropy of mixture ϕ_m. As this depends solely on the logarithms of the free paths, it depends, in the case of two gases at the same temperature and pressure, only on the relative volumes of the gases before they are mixed.

There is still another point to be considered. If a perfect gas is, say, expanded at constant temperature, its internal energy remains the same; but such gases as oxygen and nitrogen have a slightly larger internal energy at the larger volume, as discovered by Thomson and Joule, and this involves an increase of entropy in excess of that due to the increase of mean path; this may be called the Kelvin entropy or the Joule entropy; preferably the Kelvin entropy, as the term joule, being used to denote the unit of heat, might give rise to confusion. We may call this ϕ_x, if a symbol is ever wanted.

We may imagine a closed apparatus with entry pipe for air and exits for nitrogen and oxygen. We have then air going in at one pipe and oxygen and nitrogen coming out at two others, and heat leaking in from the outside. The conditions for permanent regime are that the internal energy of the O and N coming out must be equal to the sum of the internal energy of the air entering and the heat leaking in. If this were all, of course, we would arrange no heat leakage, and let the air come in and the oxygen and nitrogen come out at the pressure and temperature of the air. But we must get the entropy balance right, too—that is to say, that the entropy of the oxygen and nitrogen coming out must be at least equal to the entropy of the air going in, and if heat is leaking in, or if there is any other irreversible process such as free expansion going on inside, or irreversible conduction in the interchanger, the entropy of the gases coming out must be greater.

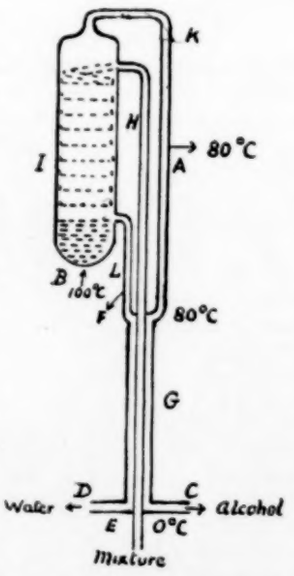


FIG. 1.—IDEAL DIAGRAM OF ALCOHOL STILL.

So many people are unfamiliar with this way of discussing such questions, that it may be better to deal with the matter also along other lines.

As the separation of the oxygen and nitrogen is carried out in a rectifier, we may discuss the action of the rectifier, taking an alcohol rectifier first as being more familiar, and not involving out-of-the-way temperatures: though, as a matter of fact, it

is in a way very much more troublesome to discuss fully.

Fig. 1 shows an ideal, or paper, diagram of an alcohol still reduced to a simple form. A mixture of 50 per cent alcohol and water at 0° C. comes in at *E* and flows through a heat interchanger *G* until it reaches 80°, the boiling-point of alcohol. It then passes up the pipe *H* to the top of the rectifier *I*. The pipe zigzags, or winds, down to a point where the temperature is 82°, the boiling-point of the mixture. The liquid is then discharged into the rectifier, which may be full of trays, and trickles down to the bottom and out by the pipe *F*, and thus to the interchanger *G*. The alcohol vapor comes off by the pipe *K*, and is nearly all condensed at *L* and passed to the interchanger. The water and alcohol come out at 0° C. at *D* and *C*.

The rectifier must have heat supplied at *B* to boil off enough steam to rise against the trickling fluid, and, assuming all the parts of the apparatus heat tight except where arrows are put, we have coming out of the system heat at *A*, where the alcohol vapor is condensed, and at *F*, where the water coming out at 100° is cooled to 80° before entering the interchanger. As alcohol and water heat on mixing, the output from *D* and *C* have higher internal energy than the ingo at *E*, but except for that, all the heat that goes into the interchanger in the pure water and pure alcohol goes into the mixture and goes back into *I* by the pipe *H*.

Suppose the still deals with 1 kg of mixture per minute, 500 grams of water come down the pipe *L*, so the heat given out in cooling the water to 80° is obtainable. The heat of mixture of water and alcohol gives the extra internal energy of the output. If the heat capacity of the mixture were the same as that of the alcohol and water separately, the water and liquid alcohol entering the interchanger would just heat the mixture to 80° as they cooled to 0°, but, as might be expected from their mixture evolving heat, the heat capacity, in which is really involved the disgregation energy, is higher, so either the water must be admitted above 80°, or the alcohol must not be all condensed before entering.

As we know the heat of vaporization of alcohol, we get the heat we could get out at *A*. Knowing the quantity of heating coming out, we know the heat put in at *B*. It might be thought that this would be anything the workman liked; this is not so, for if we put in too much, water would come out with the alcohol, and if too little, alcohol would come out with the water.

The broad result is that heat has to go in at 100° and come out almost wholly at 80°, the heat being roughly the heat of vaporization of the most volatile liquid.

To see what happens when we do the separation by mechanical power, we can imagine an ideal heat engine worked so as to deliver heat at 100° and to take in heat at 80°, or other temperature. If it does not get in enough at 80° it must make up the difference by direct degradation of work into heat or better by taking in heat at 0°; if it gets in more than is wanted at 80° it can deliver at 0°. If the process merely involved taking in the heat of evaporation of 500 grams of alcohol at 100°, and giving it out at 80°, calling the heat *H*, the engine would have to take in $H\,353/373$ at 80°, or 353° *A*, and by doing $20\,H/373$ of mechanical work on the medium, *H* would be put into the rectifier at 100° C. The $20\,H/373$ left over could be let down to 0, doing $(353-273)/353 \times 20\,H/373 = 0.0122\,H$.

Thus, if the heat of vaporization were 100, the first engine would do 5.36, and the other engine would recover 1.22, so that the total mechanical work to be done would be 4.15 of the heat of vaporization of the alcohol, or 18,000 joules per minute, or 300 watts. Instead of adopting the rough statement that we have to supply the heat of vaporization of the alcohol, we may work the example out approximately.

The alcohol vapor at *K* carries over 436,000 joules, as compared with its liquid state at the same temperature. This is not all given out at *A*, because a little is wanted for the interchanger *G*. This has mixture coming in, being heated from 0° C. to 80° C., and 500 grams of water and 500 of liquid alcohol, both at 80° C., will not heat it, as the specific heat of the mixture is higher than the mean of components. This is to be expected, because equal parts of alcohol and water give out 29,000 joules per kilogram on being mixed, so that the disgregation energy of the mixture must be greater, and the heat taken in between 0° and 80° is largely disgregation energy. The extra heat needed is about 34,800 joules. This is returned to the rectifier by the hot mixture, so we get out $436{,}000 - 34{,}800 = 401{,}200$ joules at *A*. We also get 42,000 by the water coming out at *L* at 100° C. and being cooled to 80° C. before entering the exchanger *G*. We thus have to supply 443,200 joules at 100° C. at *B*.

The heat of the water in *L* may be utilized. The difference of entropy of 500 grams between 100° C. and 80° C. is 116.5 in practical units, so the heat put in at 100° C. is $373 \times 116.5 = 43{,}500$ joules; of which 42,000 are from the heat of the water and 1500 from work done. This leaves 399,700 to be supplied by an ideal engine working between *A* at 80° C. and *B* at 100° C. Such an engine will take in 378,280 as heat and put in 21,420 as work. But there are 401,200 joules coming out at *A*, so there is a balance of 22,920. An engine working between *A* at 80° C. and 0° C. will utilize 7820 joules out of this, rejecting 15,100 at 0° C. We thus have $1500 \times 21{,}420 - 7820 = 15{,}100$ joules to put in mechanically, or 252 watts.

One of the chief difficulties in this sort of calculation is the uncertainty as to the numerical data. This case is also complicated by the internal energy of the alcohol and water separate being greater than that of the mixture.

To get rid of the effect of the internal energy of the separated liquids in the last case the arrangement shown in Fig. 2 may be discussed. Here mixed alcohol and water as vapor comes in at *O*. It goes to an interchanger *G*, where it is cooled down to 80° C., being thus condensed, and it flows by the pipe *H* to the rectifier, which is as before, except that it gives out steam at *L* instead of water. The alcohol coming out at *K* is taken away as vapor. The mixed vapors condensing in *G* will give off much more heat than is necessary to raise the alcohol vapor from 80° C. to 100° C., so heat will be given off along *G*.

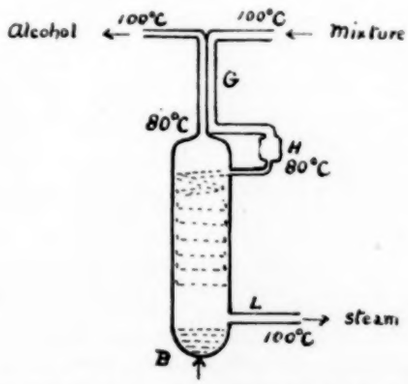


FIG. 2.—IDEAL DIAGRAM OF ALCOHOL STILL.

For want of data as to heat of vaporization of mixtures of alcohol and water I have taken the heat that would be given out by 500 grams of each and deducted the extra internal energy, 29,000. This gives 1,541,550 joules given out along *G*, after deducting 950 for heating the issuing alcohol vapor. We must therefore supply this at 100° C. at *B*. As this large amount of heat is given off all along *G* it is difficult to know its value. To get it we would need a table giving the boiling-points of the range of mixtures of alcohol and water, and the composition of the vapor coming off, and to be quite accurate we should have a column of heats of vaporization.

For simplicity we shall not be far wrong if we take the heat as being given out at 94° C. This needs 25,200 joules given mechanically and 1,541,550 taken in as heat from *G*. Another engine will have to put in 1120 joules as work, taking in heat at 0°, and giving out 3805 at 100° C. We have thus 26,320 joules per minute, or 438 watts. This figure is most likely rather inaccurate, as the data are doubtful. The example is complicated by the heat of combination of alcohol and water

too. Broadly speaking, what has to be done is to put in heat at the boiling-point of the less volatile liquid, taking out heat at a temperature between it and the boiling-point of the more volatile liquid. This involves either supplying the system with heat at one temperature and recovering some of it at a lower temperature, or supplying energy mechanically.

The case of separating oxygen and nitrogen is simpler, as there is no development of heat on mixing the gases. It is exceedingly unlikely that there is any liberation of heat on mixing the liquids, but in the ideal case discussed next, if there were such heat it would not affect the reasoning, as its influence would cancel out.

Suppose for simplicity we are operating, say in Dis, at a temperature of 90° A, and we have a system as shown with a pipe leading in air at 90° A, and two pipes delivering oxygen and nitrogen at 90° A.

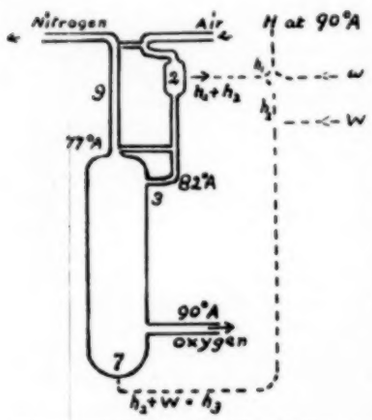


FIG. 3.—DIAGRAM OF SEPARATION OF OXYGEN AND NITROGEN.

Air comes in at *A* and the pipe branches, most of the air going to the cooler 2, where it is reduced to say 82° A, a good deal being liquefied. The partly condensed air then goes into the rectifier at 3. Nitrogen comes out as gas at 4, at 77° A, and is raised to 90° A by a portion of the entering air going down the interchanger 9. Heat has to be supplied to the rectifier at 7 to boil off the oxygen, and heat is given out at 2. Thermodynamic engines take in heat from 2, and deliver some of it at 7 and the rest to the outside. We have to find the power necessary to drive the engine to separate a kilogram of air per minute, or the energy to separate a kilogram of air. A kilogram of separated air in an ideal engine with semipermeable pistons could do 13,180 joules in allowing the gases to mix at 90° A; 13,180 joules must therefore be done in separating them. As the internal energy of the separated gases is the same as that of the mixture, this work must be wholly degraded into heat. The system that takes in air and gives out oxygen and nitrogen at the same temperature and pressure must therefore give out heat, taking in work; and the heat for each kilogram of air is thus 13,180 joules.

We do not really need to trouble ourselves as to what goes on inside our closed system. We do not need to know anything about the heat of vaporization, boiling-points, or heat of mixture; all we have to know is the work that can be imagined as obtainable by a suitable arrangement for mixing the gases, and we then know the mechanical energy ideally necessary to separate them, assuming the system to have no irreversible process going on inside.

But it is worth while to look inside.

Call the heat given out at 90° A, H, and 90° A, h, and the temperature of 2, Θ_2. Let the heat H be given out by a thermodynamic engine taking in work w_1 and heat h_1 from 2 at Θ_2. Then H is made up of $h_1 \times w_1$ and $h_1 = H\,\Theta_2/\Theta_1$, and $w = H\,(\Theta_1 - \Theta_2)/\Theta_1$. The rest of the heat given out at 2 is taken in by a second thermodynamic engine at Θ_2 and given to the rectifier at Θ_1. Call this heat h_2 then the heat given to the rectifier is $h_2\,\Theta_1/\Theta_2$ and the work done by the second engine, W, is $h_2(\Theta_1 - \Theta_2)/\Theta_2$. But $H = W + w$ and $h_1 + h_2 = h$, as the rectifier takes in and gives out the same amount of heat. From these equations we get $w = H(\Theta_1 - \Theta_2)/\Theta_1$, and $W = H\,\Theta_2/\Theta_1$.

Suppose that instead of working at 90° we lead the inlet and two outlet pipes through an interchanger, and work at 273° A, or say 270° A, to make a round number. The work W is not altered, as the conditions of that engine are not interfered with; but the first engine that takes in heat h_1 at Θ_2 now has to give it out at 270° A instead of 90° A. It is simpler arithmetically to imagine a third engine taking in the heat H at 90° A and giving it out at 270° A. This engine does 2 H.

It is hardly necessary to repeat that the mathematical power needs vary as the absolute temperature, as the work ideally obtainable on mixing the oxygen and nitrogen at atmospheric pressure varies as the absolute temperature.

Take now the case of dry air to be separated completely into oxygen and nitrogen. The work ideally obtainable from a kilogram of the gases mixing is 40,000 joules, so to separate a kilogram a minute needs 666 watts in a perfect machine where all the processes are reversible. This would come out fairly expensive on a very large scale.

In this case it has been assumed that the air is to be separated completely into pure oxygen and pure nitrogen. It does not follow that it pays to separate in this way unless both gases are wanted pure in the proportion of four to one. If oxygen alone is wanted, it does not pay to make pure nitrogen for that means abstracting oxygen from a mixture with very little oxygen, while air is obtainable at no cost. To separate a kilogram of air at 0° C. into pure oxygen and nitrogen needs 40,000 joules to be converted into heat*; to take out half the oxygen needs only 17,000 joules, but as this gives only 115 grams of oxygen, 34,000 joules are needed to give the same weight of oxygen. If the oxygen is wanted for enriching air for combustion purposes, it is cheaper to extract it impure. But for pure oxygen the most economical proportion to extract —on paper—is about 50 per cent. In practice it may be better to take out a little more, as the less gas you pass through the apparatus the smaller it may be.

It might be said that given a kilogram of air cooled down, a machine that extracts all the oxygen does its work better and is cheaper; this is not the case, however, as air does not cost anything appreciable, even cooled.

We have therefore the problem of taking in heat at a temperature of about 82° A and giving it out at 90° A, and the temperature of the room, say, 290 A.

There are broadly two methods in use. It is not necessary to trace the history of the liquefaction of gases, and to follow out the ingenious intensification methods. They are all well described in the literature of the subject.†

The first method is Linde's. This takes advantage of the discovery by Kelvin and Joule that such gases as air have less internal energy when compressed. If you feed a heat insulated system with air under compression and let it expand before it comes out, the air coming out must at first take out more energy than the air going in carries with it; so the inside of the apparatus must get colder and colder. If the outcoming air comes through an interchanger, it will come out at the same temperature as the ingoing air, so the other end of the apparatus must get colder and colder until some of the air is liquefied. The balance will come when the internal energy of the gas and liquid air taken out is equal to that of the compressed air going in.

In practice the same machine does the liquefaction and separation; but it is simpler for our purpose to treat a machine in which air is compressed and allowed to expand as a thermo dynamic engine, using the liquid air as a means of extracting heat at a low temperature. It was this invention of Linde's that made the production of liquid air, and his subsequent invention of combining the rectifier with it that made the commercial production of oxygen thermodynamically from air possible.

* For simplicity, atmospheric pressure is taken as a megadyne per square centimeter; and heat is taken in joules, not calories; entropy, when necessary, can be taken in entrops, which, multiplied by the lowest available temperature, give the incurred waste in joules.

†The engineer will find G. Claude, "Air Liquid," and Ewing, "The Mechanical Production of Cold," interesting.

The other invention is due to Claude. It was, of course, obvious that cold could be produced by compressing the air at a high temperature and then expanding it so as to do external work at a low temperature, and the merit does not here lie in the idea. But to work out such a system is a very different matter. The practical difficulties in making an engine that would give out any reasonable power at a temperature of 80° A are very serious indeed.

So far I have discussed the subject on what are sometimes incorrectly called theoretical lines, that is to say, ideal lines, in which all sorts of assumptions are made so that the discussion should be easier. Thus I have discussed perfect heat insulation and completely reversible changes and perfect thermodynamic engines. The principles are the same on a large scale; but there is a leakage of heat, and all sorts of irreversible changes of heat, and a compression pump does not come up to the thermodynamic ideal in efficiency. In the industry as now carried on the question of efficiency is of very little importance. Oxygen is sold in small quantities in bottles; and the costs of compressing, handling, selling, delivering and so on, and the collection of the empties, are enormously large in proportion to the mere cost of making. Thus we have discussed oxygen, taking 34,000 joules for 230 grains, or about 2 kw-hours per cubic foot.

The great future for the separation of the gases is in the direction of making nitrogen for the manufacture of calcium cyanamide, and oxygen for use in blast furnaces, and other work where a high temperature may be worth the extra cost. It is, therefore, important to see how far the gases can be produced on a large scale at prices within a reasonable margin.

We may, therefore, sum up the rectifying part of the plant as an apparatus which gives up heat at a low temperature, say 82° A, and absorbs heat at, say, 90° A, the heat taken in at 90° A being equal to that given out at 82° A. A refrigerator must, therefore, be a thermodynamic engine capable of giving heat at 90° A, absorbing heat at 82° A and giving out heat at the temperature of the works, say, 290° A.

It is generally supposed that Linde depended on an irreversible expansion of a gas. This is not the case, and I would put the matter in another way.

If gas is compressed, cooled to, say, 290° A, and sent into an interchanger, out of which it comes by a return pipe at the pressure of the air, there must be cooling going on inside, as the air delivered has higher internal energy than the air going in. You have therefore an apparatus for producing cold—that is to say, for absorbing heat at a temperature below the room. The question is, what is the lowest temperature at which this apparatus will take in heat? The lowest temperature is that of the boiling-point of the liquefied gas at the pressure of the atmosphere. If the apparatus is working in such a way that it can take in heat H at temperature Θ_2 the increase of entropy is H/θ_2. But if instead of working in this way the heat is taken in at Θ_1, which is a higher temperature, the increase of entropy is H/θ_1, which is less, so the apparatus makes up the deficit of entropy by irreversible expansion.

On starting up a Linde machine it begins by abstracting heat from its own air and metalwork at the temperature of the air, so the entropy is made up by irreversible expansion. As the temperature at which the heat is absorbed falls, the irreversible expansion gets less and less, until finally the whole of the gas is turned into liquid. Then there is some production of heat by the liquid passing from a high to a low pressure through an obstruction, and also some slight evolution of gas to cool the released liquid from the boiling-point under the high to the boiling-point under the low pressure. The irreversible loss due to liquid is $pv/10$ in joules, where v is the volume of the liquid. If the compression is small, this is small, but if it is high, pv may be considerable. It would be quite easy to get this energy out by means of a little engine, as there is no expansion and no difficulty in lubrication, and it would be worth while on a large scale. The liquid may also be cooled to a lower boiling-point while still under pressure, so that there is no irreversible evolution of gas on its being exhausted from the engine.

The refrigerator has to do the work of two thermodynamic engines, for it has to abstract a lot of heat at one temperature, give a large proportion back at a higher temperature, and give out heat at the temperature of the works. The ideal apparatus consists of a compressor with water-cooling delivering gas at the temperature of the works and a high pressure to an interchanger. The pressure is chosen so that the gas liquefies under it at 90° A, giving up its heat to the rectifier. The liquid at 90° A is then cooled in an interchanger to 82°, and it is then passed through a hydraulic engine, coming out at atmospheric pressure and temperature 82°. It is now vaporized, taking in heat at 82°, and goes out by the delivery side of the interchanger.

The process described is not quite reversible, because the specific heat under constant pressure of the gas coming down the interchanger is greater than that of the gas coming up under atmospheric pressure; there has therefore to be an abstraction of heat all along the interchanger to make up for this difference.

The idea of this apparatus is something like that of an ordinary carbonic acid machine worked under different conditions. If a carbonic acid machine were worked in the tropics and the cooling water were above the critical point of CO_2, the compressor would supply compressed gas only. Some sort of refrigerator would be necessary to liquefy the CO_2, and the liquid CO_2, would then be evaporated at a low temperature. We thus have three temperatures to consider—the temperature of the works at which heat due to compression is given out, the temperature at which the CO_2 is liquefied under pressure and the temperature at which the CO_2 is finally evaporated. In a refrigerating plant three temperatures would be a nuisance, so a fluid is chosen with a critical point above the temperature of the works. But for rectifying three temperatures are necessary—the temperature of the works, the temperature of the boiling-point of the least volatile liquid and the temperature of the liquefaction of the mixture. We can therefore use a fluid which is compressed at a temperature which may be much above the critical point of the liquid.

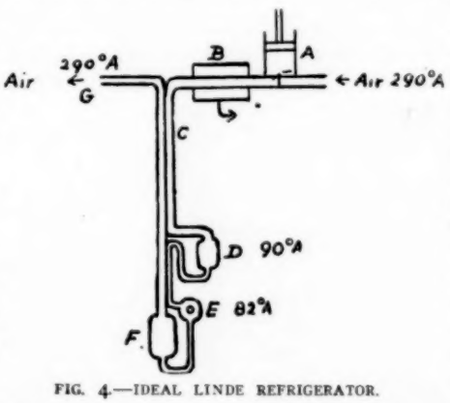


FIG. 4.—IDEAL LINDE REFRIGERATOR.

Fig. 4 shows an ideal Linde refrigerator. The compressor A delivers a fluid, say air under pressure, to the cooler B. Heat comes out there at the temperature of the works. The air goes on down through the interchanger C until it is cooled to 90° A. Then it is cooled in D, which is really a coil of pipe immersed in the oxygen to be evaporated at the bottom of the rectifier. From D the liquid, still under pressure, goes down the interchanger until it is cooled to 82° A, and it then works a little engine, E, and comes out into F. F is really a coil of pipe in the liquefier of the rectifier. The air is evaporated in F, and passes up the interchanger and escapes at G.

There are some points to be noticed. As already pointed out, the interchanger must take in heat by leakage, which is an irreversible process; and the heat taken in at F will be greater than that given out at D, as it takes more heat to vaporize a liquid at a low pressure and temperature.

The processes in this refrigerator are, however, substantially reversible.

If schemes for dealing with liquefied gases are examined, it will often be found that a liquid is being condensed or evaporated by a gas in the interchanger. That is always a piece of bad design, involving irreversible change. It may be inevitable, of course; but it should be recognized as an evil which must be carefully watched.*

Linde points out that the mechanical work necessary may be reduced by working between two high pressures. But there is a limit to this, as if both pressures are very high the lower temperature limit of the plant is reduced; so the limit is reached by the lower pressure, which must be low enough for the fluid to evaporate at 82° A, or whatever temperature the heat has to be extracted from the rectifier. If we choose a fluid, say nitrogen—this gives the lowest pressure that can be used. Then the other pressure must be high enough for the liquid to condense at 90° A. Some latitude can be got by control of the pressures in the rectifying plant, but it is simpler for the present to take the rectifier as working at atmospheric pressure for the sake of ease of calculation.

The refrigerator, then, has to supply for each kilogram of air split into pure gases 135,000 joules at 90° A, absorbing 135,000 at 82° A. The liquefying of this fluid is in contact with a fluid that is being evaporated, so that that part of the process is ideally reversible. The re-evaporation is in contact with the air which is being liquefied, so that part of the process is ideally reversible. The passage of the liquid from the high to the low pressure can very easily be made to give out work. It is not the same problem as dealing with expanding air, as the volume is small, the pressure constant and the liquid acts as lubricant. The process, so far, is essentially reversible. The specific heat of compressed gas is higher than that of the same gas at a lower pressure, so a little allowance must be made for that. It is thus possible to use a process which has no external work done by expansion of cooled gas in such a way as to get little loss by irreversible changes.

To work out an example, nitrogen can be taken as the fluid, as it is easier to discuss than a mixture. To get 135,000 joules, taking heat of vaporization of nitrogen 200 joules per gram, we need 675 grams of nitrogen, and we can liquefy this at 90° A under about four atmospheres, evaporating it again at 82° A at two atmospheres. Taking the density of liquid nitrogen as 0.8, the hydraulic engine gives out 170 joules, a mere trifle in this case, and the compression of the nitrogen from two to four atmospheres needs 37,000 joules, so we need under 37,000 joules. As the separation needs 40,000, this is clearly an erroneous figure, but the data are very uncertain. We must need more than 40,000.

To work on a large scale with atmospheric pressure in the separator and four and two in the refrigerator would be quite out of the question. The volumes of air would be enormous, and the interchangers would be very inefficient. Those who have had to work with gases that have to be heated or cooled will realize the extent of this trouble at once. To make the plant practical it will be necessary to work the rectifier under pressure. This raises the temperature of the rectifier; but that does not mean that we can separate the gases more economically; the cost of separation depends, as already stated, on the difference of entropy of the gases entering and leaving the plant, not on what goes on in the plant, provided there are no irreversible changes. Using pressure allows the refrigerator to be worked at higher pressures, so that the interchanger may become practical.

Assuming the ideal energy needed to separate a kilogram of air into its gases to be 40,000, the question is: What will it cost in practice? On a large scale it may be safe to say 100,000 joules. We would then get 230 grams for 100,000 joules; or a ton would need 120,000 kw-hours. Taking the cost of a kilowatt hour as low as 0.2 cent, this means $250 a ton.

It does not look as if oxygen at this order of figure could be used for blast furnace work. Taking a blast as using four and a half tons of air per ton of iron, to increase the oxygen content of 1 per cent of the oxygen content the cost comes out at $2.50 per ton of pig. This does not look as if it could be of use.

The principle of separating gases by cold can be applied to other gases. Thus hydrogen may be purified, so that water-gas can be used as source of balloon gas. Methane can be separated out by solution in water under pressure.

Chlorine can be separated from the impure gas from the anode chambers in electrolytic works. Such chambers are generally worked under a slight exhaust to avoid troubles due to leaks. Air thus leaks in, and the chlorine coming off is moist and impure. Chlorine can be condensed into water, and it might appear that condensing such gases as chlorine and methane into water would allow of their being separated with less expenditure of energy than the entropy consideration demands. This is not so. It takes greater pressure to get the gas into the liquid out of a mixture than the solution exerts in an atmosphere of the pure gas.

Chlorine, by the way, is easier to handle than is generally supposed. It can be pumped in iron pumps flooded with sulphuric acid. It can also be dried and then compressed in iron pumps lubricated with sperm oil saturated with chlorine. It is not necessary to dehydrate it absolutely for compression up to, say 50 lb.

There may be other gases that can be separated or purified by liquefaction and rectification; it is difficult to think of any case where the principle cannot be applied.

* There is another fallacy very common with regard to gas issuing from a nozzle. Suppose it comes out into the atmosphere, it is thought that as it has to expand against the pressure of the atmosphere, doing work in joules equal to a tenth of its increase of volume, it must be cooled to a corresponding degree. This is a mistake nearly everyone makes until he thinks of the matter a second time. As such an error would give rise to many miscalculations in connection with the liquefaction of gas, it is not impertinent to call attention to it.

Progress in Ore Dressing in United States and Mexico During 1910.

By H. A. Guess.

Water Concentration.

The record of progress during 1910 compares favorably with that of any recent year. If it were possible to make a general observation outlining the direction the various betterments have taken it could be said that progress and improvement had been made chiefly in the way of detailed steps looking toward the recovery of the mineral, without unnecessary sliming—either by stage grinding with mills, with recovery of the released mineral before the second grinding, or by carrying roll recrushing to such a degree of fineness that in certain cases it largely replaced the work of grinding mills. Coupled with this has been the attention devoted to the better classification of table and vanner sizes and especially to the more complete removal of all slime from feed going to stratifying tables. Equally important, too, have been the little changes and betterments—bulking large in the aggregate—which have been going on at the different plants over the country, all tending to economy in operation, the handling of a larger tonnage, the making of a cleaner product or a higher recovery.

Following the ore in water concentrating processes by steps it may be said that nothing noteworthy in the way of new machinery or arrangement has developed during the year in coarse or dry crushing preparatory to concentration.

In jigging—where jigging is used—the Hancock jig, with its large capacity, has continued to find favor, particularly in the lead district of Southeast Missouri, and all mills built there within the past two years use that type of jig, some 9000 tons per day being at present handled there by that system.

In screening other than for sizes finer than 1 mm, trommels continue to be the more usual equipment, but of late the impact screen, which after its earlier trials of some ten years ago has fallen rather into disrepute, has been improved mechanically so that its use is greatly increased, some of the largest of the

concentrators on porphyry copper ores now adopting impact screens entirely.

For fine crushing the tendency has been to carry the work as far as possible with rolls, and where grinding mills are used it has been in many cases found profitable to conduct the operation in two stages with classification and table treatment in between, the coarser sizes being returned from the roughing tables to the second mill. Certain of the porphyry concentrating plants have adopted and within the past year extended this system with improvement in recovery, as well as increase in tonnage per mill section and per horse-power consumed. During the year extended working tests were carried out under the above-mentioned system in comparison with direct crushing with Nissen stamps and the advantages of the stage-grinding system, in such comparison, were clearly apparent in tonnage output per square foot of mill floor space, in power consumed and in percentage recovery. Mills of the Chilean type appear to find most favor, especially among the larger plants, although altogether there are doubtless more Huntington than Chilean type mills in use throughout the country. Occasionally tube mills are seen in concentration plants for fine grinding of certain middling products, and their use within the past year has been on the increase, particularly the more recent Hardinge conical type. The tube mill has heretofore been regarded chiefly as a slimer, due to its extended use for that purpose in cyaniding, but by suitable adjustments in rate of feed and ratio of water it is showing encouraging results as a fine granulator in comparison with the more usual grinding mills, the percentage of slime formed being not excessive nor the horse-power and supplies unduly high.

The classification of table and vanner sizes has received much attention during the past year, especially in regard to the more complete removal of all slime from feed going to stratifying tables. The removal of slime is not difficult in the coarser sizes of table feed, but in the finest sizes it has been impossible by any continuous discharge type of hydraulic classifier to make the separation from slime clean cut. The usual procedure in such cases has, therefore, been to recover as much of this slime as possible by collecting the table back-water for subsequent vanner treatment and to let the rest go to waste with the table tailing. In cyanide practice mechanical classifiers are largely used for the separation of fine sand from slime and the same idea is now beginning to be applied at some of the larger water concentrating plants. At the largest plant in Southeast Missouri mechanical classifiers similar to the Esperanza drag classifier of cyanide practice, only a little longer and arranged with spray pipes for washing, have been installed—one in each 600-ton mill section—taking the undersize of 1-mm trommels and separating therefrom all slime as overflow product and all sand as a washed drag product, the slime after thickening going to vanners and the sand going to hydraulic classifiers for division into various sizes of table feed. Another system lately adopted on a large scale by certain of the porphyry copper concentrators, and looking to the same elimination of slime from all table sizes, is the use of intermittent discharge hindered settling classifiers of the Janney type, which deliver the classified products in thickened form. The Richards vortex and hindered settling classifier and also the more recent pulsating classifier continue to have a wide use.

In table concentration each year sees new models—all practically of the original Wilfley type—put upon the market. Joplin was about the last of the older established districts to awaken to the losses being made through inattention to the treatment of sizes too fine to jig and in this and its adjoining districts many tables have of late been installed as adjuncts to the older jigging plants. A large proportion of these are of local manufacture, some half-dozen different styles being on the local market there, all of them of very good construction. In the other and better known districts of the United States and Mexico the Wilfley, Overstrom, Card, Deister and James continue to be the types more frequently seen and in about the order mentioned.

The Deister No. 3 table competes fairly successfully with the vanner on slime, its merits being more or less confirmed in that one of the large porphyry copper concentrators now nearing completion is adopting the Deister No. 3 for its slime concentration. In general, however, for slime concentration vanners continue to be the standard machine, although there is recently some revival in the use of buddles on slime, the larger installations being of steel frame with concrete top; also some interest has been aroused in traveling canvas-belt machines, such as the Craven slimer, which in the matter of recovery does good work. It is usually considered that in vanners the Frue type does a little better where there is considerable mineral, the Johnson suspended type being more in favor for second treatment or where the mineral is in small amount. The Isbell vanner supported from a steel girder foundation, instead of the more usual four-post construction, has been introduced this year and may in practice present some features of improvement. Retreatment of vanner tailings over a second series of vanners is now becoming accepted practice unless the ore is very lean, the first tailings being usually thickened before going to retreatment.

The disposal of tailings is, with the advent of larger mills, becoming something of a problem, especially in a flat country where surplus water is not available. In Missouri, where such conditions exist, the more recent plants use dewatering devices feeding the comparatively dry tailings to an inclined conveyor belt which acts as a stacker, the motor drive being at the upper end. Extensions to the belt are made as occasion demands, the upward slope being maintained—usually about 16 ft. to the 100 ft. It may be noted that for such work the less troughing the belt receives the less tendency to crack and the longer its life, and this in less degree is also proving the experience in the handling of dry material with conveyor belts.

Cyaniding.

In cyaniding the practice of all-sliming treatment is on the increase and the tube mill remains the approved machine for such work, although at some of the larger plants comparative tests are still being run of tube mill versus pan, for fine grinding and sliming, and at the Homestake a number of Wheeler pans are in regular service. In connection with the tube mill as a slimer a comprehensive series of grinding tests at Pachuca —details of which were given in a paper by Mr. V. B. Therrod before El Instituto Mexicano de Minas y Metalurgia—led to the conclusion that the grinding efficiency increases with the percentage of solids in the feed up to 55 per cent to 60 per cent; that it also increases with the quantity of ore fed per minute to a point which varies with the dimensions of the mill, and that the horse-power efficiency appeared to be lower in the mills of larger diameter—5 ft. x 20 ft.—than in the smaller mills—4 ft. x 18 ft.

In the separation of sand from slime for purposes of return for regrinding or for leaching, the Dorr classifier continues to have a wide use, and more recently, to some extent, drag classifiers of the Esperanza type. For the same purpose the Caldecott diaphragm cone is also being adopted, and its use seems on the increase.

For the agitation of slime preparatory to filtration the Pachuca tank or its equivalent continues to be the accepted system for new installations, and at several plants these tanks are of late being connected up in series, the first overflowing to the second, and so on, thus giving continuous agitation in transit without loss in head; with simplicity in operation and equal percentage extraction. Considerable interest has been aroused by the Just process, which uses agitation by air through porous silica brick placed as a floor in flat-bottomed tanks, the air being introduced beneath and coming up evenly through the brick. The same material is also being used to some extent as a filter in leaching tanks.

Much attention during the year has been devoted to the development and perfecting of filtering processes for slime. Various patented filters, both vacuum and pressure, have been

added to the already large number of such devices on the market, so that, with intelligent selection of the machine best adapted for each particular type of slime, steady progress seems assured in the matter of filtration. In this connection, too, the use of slime thickeners of the Dorr type is relieving the work of filtration to some extent. Of the various filtering devices now on the market the best known are, perhaps, the Butters, Merrill, Burt, Grothe-Carter, Ridgeway, Kelly, Moore and Blaisdell.

In precipitation the economy of the zinc dust over the shavings method in labor and supplies is such that its use is being more widely adopted each year, especially at the larger plants and new installations.

Altogether, it may be said that each year sees great increase in the variety of ores successfully treated by cyaniding, and especially the silver and silver-gold ores of Mexico.

Miscellaneous Processes.

The more usual ore-dressing processes are the foregoing, namely, water concentration and cyaniding, either of which may sometimes include amalgamation.

Of the remaining processes it may be said that chlorination continues to have a limited application in the treatment of certain concentrates and also continues to compete with cyaniding in the treatment of certain crude ores, notably in Colorado; that leaching of certain copper ores is still occasionally practised, using either dilute acid or acid ferric salts; that a few dry concentrators of the S. & S., the Behrend and other types are being installed now and then, where no water is available, and that recently the Quinner dry-placer machine is being used to some extent, as, for example, in the Altar desert district of Mexico.

In general, however, it may be said that the special processes, for the most part, center around the recovery of zinc blende from complex materials, usually middling products from wet concentrating mills, containing in addition iron and perhaps copper sulphides, carrying usually some gold and silver, frequently also with small percentages of lead remaining from the wet concentration, some of the gangue material being often of heavy specific gravity—garnet, siderite, barite, fluorite and the like.

Considerable progress in the separation of such materials has been made during the past year, the processes grouping themselves, in the main, under three heads—magnetic, electrostatic and flotation.

In point of tonnage handled the magnetic processes of separation continue to lead, either with or without preliminary roasting; in the former case the iron and copper being the magnetic products, and in the latter usually the zinc. The Wetherill, both belt and rotary, the Dings and the International are the more usual types in use.

The electrostatic process is now represented by the Huff separator, which after being perfected in the separation of zinc and iron sulphides in Wisconsin, has shown good results similarly in Utah.

In flotation the Elmore oil-vacuum process is now giving very fair results in one or two small installations in Mexico, the grade of zinc recovered being particularly good. Its application in this country, however, has so far proved limited.

Several of the various acid and oil-acid flotation processes as used at Broken Hill, including also the Horwood process, have been tried out in America, but so far nothing has developed beyond the experimental stage.

Without doubt the most interesting of all the flotation processes so far is that of the MacQuisten tube, in which the sulphides are mechanically floated upon water by passing through tubes some 8 in. in diameter by 6 ft. long, riffled internally by a rib of triple or quadruple pitch. In point of size 20-mesh material is about the maximum that can be readily floated, elimination of slime and a certain amount of classification being also required. A disadvantage is the small capacity per tube, about 3 tons per 24 hours, and the necessity of passing the material through a series of tubes—perhaps a maximum of six in all—in order to obtain a satisfactory extraction. The chief field would seem to be where water concentration fails, as in the separation of sulphides from gangue of about equal specific gravity—such as garnet, siderite, etc.—and very good results are now being obtained upon such material either as original ores or middling products.

Flat River, Mo.

A New Electric Furnace for Steel Melting and Refining.

By Joh. Härdén, E.E.

The electric steel furnace appears to be slowly but steadily gaining ground in this country (England).

A 1½-ton Kjellin induction furnace has recently been erected in Sheffield for crucible steel work, and has successfully completed its guarantee run. This furnace is especially adapted for producing the highest grade of tool steel and is regarded as a most effective adjunct to a crucible steel plant. It is, however, well known that a furnace of this type is not adaptable for any extensive refining of steel, owing to the refining surface of the bath being small.

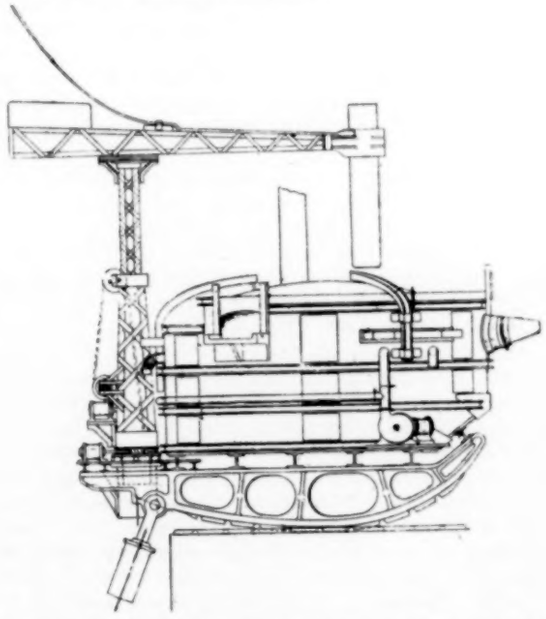


FIG. I.—VERTICAL SECTION OF COMBINED ARC AND RESISTANCE FURNACE.

It has long been recognized by steel makers generally that although the induction furnace is an excellent tool in the hands of the crucible steel maker a furnace which would permit more extensive refining, and also the construction of large units, at the same time keeping the advantages embodied in the induction furnace system, is much to be desired.

It is because of the comparatively low power factor of larger furnaces of the plain induction type that this class of furnace has not been constructed in large units to any great extent.

In order to overcome this the Roechling-Rodenhauser furnace, which is partly a resistance furnace and partly an induction furnace was designed. A considerable number of these furnaces have been erected and are in operation abroad, giving very good results.

Many steel makers, however, are of the opinion that it would be an advantage if a furnace could allow superheating the slag for refining purposes, but at the same time enable the metallurgist to so regulate the heat of the charge that gases could be expelled without difficulty. At the same time the

perfect circulation of the bath should be maintained, so as to secure a perfectly uniform material. A new furnace designed on these lines, which the inventors call the "paragon" furnace, appears to possess many points in its favor.

As is shown in illustrations, Fig 1, 2 and 3, this furnace is a combination of an arc and a resistance furnace, which makes it very much like an open hearth furnace, and it may indeed be worked exactly in the same manner.

The slag blanket is kept at the desired temperature by means of arcs playing over the surface of the slag while the remainder of the heating power is transmitted to the bath by means of terminal plates, such as are used in the Roechling-Rodenhauser furnace.

The result of this combination is that the slag may be heated to any desired temperature while dephosphorization and desulphurization take place, but the heating effect of the arcs need not be increased to such an extent as to keep the whole charge superheated, which, of course, would tend to overheat the steel immediately beneath the arcs, but a carefully regulated amount of current can be at the same time sent through the resistance plates, thus also heating the charge from underneath.

When the dephosphorization and desulphurization are finished, it may be desirable to alter the gradient of heat in the furnace so that the lower portion of the bath is hotter than its surface, in order to expel the gases more readily. This result is easily obtained by diminishing the power transmitted through the arcs and increasing the power transmitted to the side plates. In fact, it is even possible to extinguish the arcs for a time and keep the bath liquid by means of the side plates, which for this purpose are provided with a cooling device of enforced air.

It is obvious that the metallurgical process is thus considerably facilitated, inasmuch as the metallurgist has the

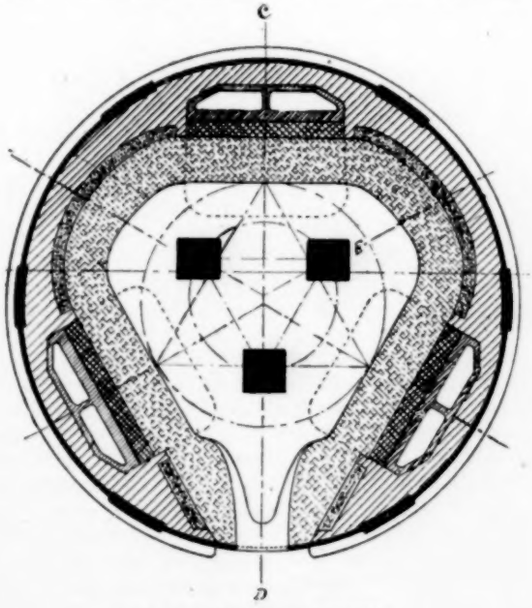


FIG. 2.—HORIZONTAL SECTION.

power of controlling the gradient of heat in the bath at will.

This combination has another advantage, viz, that the electrodes do not require to be of such large section for a given capacity of furnace as would be the case if the total amount of power were transmitted by means of the arcs only, owing to the fact that normally about 50 per cent of the total power is transmitted through the arcs, while the other 50 per cent is transmitted through the side plates. The advantages, therefore, are threefold.

Firstly, the size of electrodes for a given furnace capacity is only about half that of an arc furnace, and consequently with a given electrode section available the melting capacity of the furnace can be nearly doubled.

Secondly, owing to the destructive effect of large arcs immediately under the roof of the furnace, the roof very quickly deteriorates, but in the case of this new furnace the roof is not subjected to such severe strain owing to the reduced size of the electrodes and to the smaller amount of power transmitted through the arcs, therefore the maintenance of the furnace costs less and the roof lasts longer.

Thirdly, the control of the load factor of the furnace is

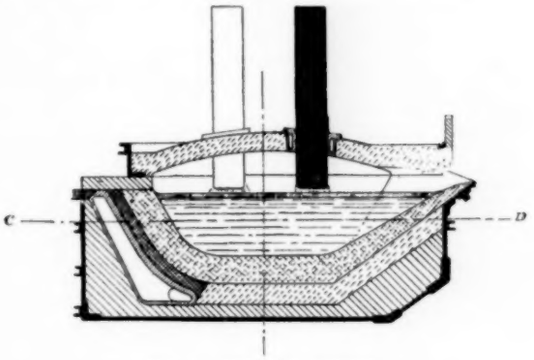


FIG. 3.—VERTICAL SECTION.

better because the current flowing through the resistance plates is a perfectly steady one, whereby the heavy current fluctuation otherwise caused by large arcs is compensated.

The result of this is that no special machinery is required either for compensating for the lower power factor of a true induction furnace or designed with a large flywheel effect, as in the case of a simple arc furnace, on account of its fluctuating load factor. Standard machinery and standard transformers only appear to be necessary, which, of course, decreases the first outlay for the plant, and under the circumstances may prove to be almost a deciding influence.

The power consumption per ton of steel should also prove more economical by means of this combination, as the high temperature of the steel is only required during desulphurization. it being well known that dephosphorization does not require such a hot slag as desulphurization, and it follows as a logical sequence that the higher the temperature of the steel for any length of time the greater also the heat losses during the process; in view of this, the total melting and refining process should be more enconomical with this combination.

It, therefore, appears that the type of furnace includes several of the advantages of the induction furnace, viz, a good circulation of the charge, inasmuch as it is heated from below as well as from above, and a high load factor.

No electrical apparatus, in the ordinary sense of the word, seems to be an element of the design of this furnace, because the power supply transformers may be situated in an adjoining building protected from dust and injury.

The low voltage current only requires to be transmitted to the furnace by means of insulated bus bars and cables.

The regulation appears to be simple, as if the furnace is operated direct from a generator the field rheostat controlling the current to the generator is all that is required; or, if transformers are used in connection with an existing power supply, so-called regulating transformers with interchangeable terminal steps are employed. In fact, comparatively little regulation in this respect should be found necessary, because the current in the arcs ought to be easily regulated by altering the length of the arcs, while the current in the resistance plates is more or less automatically regulated by the condition of the charge.

The author is informed by the owners of the patents, Messrs. The Grondal Kjellin Company (Ltd.), that furnaces of this type will shortly be tried.

Review of the Metallurgy of Copper in 1910.

By L. S. Austin.

Conveying and Transmission.

The belt conveying system at Cananea, which at first presented difficulties in operation, is now in continuous use. It is ideal in theory and it costs only half of any other effective system. R. R. Moore states that the conveying belt, as used at Garfield, has stood the wear and needed but little repair, and the system is now in satisfactory operation. It appears that when installed by engineers experienced in this special line and afterward reasonably handled the conveying belt is durable and smooth in operation. The management of the International Smelter at Tooele, where this system has been installed, is also apparently of this opinion.

In general, in feeding the blast furnaces, where the charge and fuel are weighed, the charge-car is commonly used. Of them all the Hodges car used at Granby works most rapidly and effectively. The load of the car can be adjusted to the condition of the furnace both as to quantity and position and can be dropped where desired. In those systems where the car delivers at the long sides of the furnace it is shot in more at random, and this is apt to show itself in irregularities on the surface of the charge, in more hanging up and in more overfire.

In Mexico, at the plant of the Tezuitlan Copper Company, Mr. Cyrus Robinson, the constructing engineer, convinced of the importance of a uniform charge both in the size of the ore as well as of its composition, takes every precaution to assure this. There are two lines of wire-rope tramway, one of 15 tons hourly capacity carrying limestone to the mouth of the mine, the other of 45 tons capacity taking both ore and limestone to the smelter. Storage is effected in steel bins 40 ft. high and 25 ft. diameter. There are five bins at the mine and twelve at the smeltery having a combined capacity of 3000 tons. The bin bottoms are of rocks dumped in to form the slope which is then covered by rails set at an angle of 45°. These cost but half what a rectangular wooden bin would come to. The ore is delivered to each of the bins in turn to secure even distribution. Finally it is drawn off into a large charge-car and the coke is added. There is an overhead feed-hopper of 65 tons capacity into which the car drops its load. This hopper is divided into partitions to half its depth. A row of gates handled by hydraulic cylinders enable any gate to be opened where desired, so that the ore is dropped where the furnace most needs it, thus securing more elastic feeding conditions. Ordinarily this is done by dumping as seems best into the side doors of the furnace and to some extent (though seldom done) by trimming the charge.

Roasting.

This is now largely done in one or another type of the McDougall roasting furnace. These furnaces have been improved in mechanical details. Some attempts have been made to do away with the water cooling, or at least to minimize its abstraction of heat from the charge. At one plant the rabble blades are slipped over the rabble arm and can be readily slipped off when they fail. But besides this the bladeholders have been shaped so that the flue dust lodges as a layer on them, so preventing the abstraction of heat. As a result less cooling water is used, the arms are preserved and the output of the furnace has been increased. At one of the larger plants the concentrates, carrying 25 per cent copper, really need but little roasting for reverberatory treatment. They are accordingly thrown upon the lower hearth of the roasting furnace, where sulphur is eliminated and where they become hot, and so in suitable condition for the reverberatory smelting to follow.

The dropholes of the McDougalls by which the ore falls from floor to floor are set opposite the sidedoors. Here the ore sticks to the brick work and is hard to remove. This brick surface has been replaced by iron plates and from them the accumulation can be easily barred off. R. R. Moore in his article on reverberatory practice speaks of brackish water as causing scaling of the water surfaces and pipes. The remedy for this would seem to be to use a good quality of water over and over by putting in a satisfactory water-cooling tower.

Reverberatory roasting produces more sulphuric anhydride than blast roasting, and we find the plants using the McDougall furnace successively putting in brick offtakes because of the short life of the sheet-iron outlet flue.

Reverberatory Smelting.

To get rapid melting of the charge in the reverberatory furnace we must burn the coal rapidly. But the gases escaping at the throat should be at least 1100° C. or at the melting point of the slag. Otherwise we encounter the difficulty experienced at first at the Garfield Works when the draft was poor, viz., the setting or crusting of the slag at the front end of the furnace. But with hot gases the question arose how to utilize the heat of these gases and this has been solved generally by using it to produce steam for power purposes.

In open-hearth steel practice the problem has been differently solved, that is by recuperation, by which the hot gases go to a brick checker work and upon reversal of the current air is drawn through the checker work, which adds thus to the intensity of combustion. In the open hearth the temperature is built up by a series of reversals of draft, so that care has rather to be taken to avoid so high a heat as to melt down the roof of the furnaces. It has been stated that the rate of transmission of heat to the charge or the rate of melting will vary with the square of the difference of temperature between its melting point and that of the burning gases. Thus, assuming the melting point of the charge to be 1100° C. and that of the gases 1500° C., melting will proceed at a certain rate due to the difference of 400° C. If the heat of the gases is increased by 100° C. only, or to 500° C. in all, the rate of melting will be increased in the ratio of 16 to 25, or 56 per cent. Is it not better, then, to adopt the open-hearth idea of utilizing the heat to the best advantage for the object in view rather than to employ it for an ulterior purpose? It reminds me of a similar problem in blast-furnace work. Is it better to effectively briquette flue dust or to so run the furnaces as not to make any?

An advantage possessed by the reverberatory furnace over the blast furnace is that the charge can be greatly varied in composition and yet good work be done. The work may be slower, but so long as the slag flows it will be clean. On the other hand, in the blast furnace a viscid slag eventually refuses to issue from the taphole and the furnace "freezes up."

Another advantage is that flue dust is made only at the time the charge is dropped. So little is the loss in this way that no damper is closed at the time of dropping. The passages to the main flue in ordinary practice are large and open and there is little accumulation of dust to interfere with the draft. At Great Falls, where the regenerative open-hearth furnace has been tried, it is claimed that the checker work has been partly closed by accumulating dust. This difficulty should be overcome by a design such that the dust can be removed as it accumulates, and so that the regenerators can be repaired without serious dismantling.

The tendency to-day in copper mining, exemplified in the porphyry coppers, is to handle lower-grade ores that necessarily require concentration. It is these concentrates, too fine for treatment in the blast furnace, that seek an outlet for treatment by the reverberatory. To obtain tonnage in the latter furnace its length has been increased step by step until now a furnace 100 ft. long appears to give the greatest satisfaction. To retain the heat the walls of the furnace have been increased to 30 in. and the roof to 15 in. or 18 in. thickness. So heavy a roof, however, limits the width of the furnace to about 20 ft. and smelting area must be got by increase of length.

The practice to-day seems to confine itself to dropping the charges near the firebox end and to depend upon the charge spreading out gradually owing to its tendency to flow while hot and fine. It accordingly floats slowly toward the front of the

furnace, melting as it goes and dropping out its burden of matte as fast as the matte forms. The result is a cleaner slag. It appears probable that this practice has come about because the heat was so rapidly abstracted from the burning gases by the charge that the fire end was the only effective place for the melting. What might be the effect of regeneration and at the same time of an increased melting area by dropping at other points, as was formerly done, remains to be seen.

The Coal Bed.—In Wales the importance of a deep firebed was long recognized. Clinkers were allowed to accumulate on the grate bars to furnish a bed for the live coals, and by pushing aside the grate bars those clinkers were removed and the fire kept clear. In firing the firebox was filled to the top of the bridge wall and coal was used to close the firehole, thus dispensing with a firedoor. It is along these lines that firing is yet done, generally using the old style of grate bar, which can be so well manipulated. Still, to burn coal rapidly frequent grating should be practised, and this means a loss of perhaps 10 per cent of the fuel as coked droppings. This coked material can be washed for the removal of the ash, or otherwise burned with other fuel for steam making. The depth of the bed may vary from 2 ft. to 4 ft. as experience indicates, but even the deeper beds occasionally burn with holes. The coal is dropped from hoppers and consequently does not get even distribution on the firebed. While doors have been provided for dressing the surface of the fire, it is hard to see its condition in the highly heated firebox and so air may get through to the hearth and thus cool the melting zone. Experiment with the larger furnaces shows that, especially before a new firing, 50 per cent more than the theoretical quantity of air may easily get in, with consequent loss in the intensity of melting heat. It makes one disposed to recommend the down-draft method of burning coal such as Hawley has instituted for boiler firing. Here the upper surface of the coal bed is easily seen and the chance of irregular burning is avoided. Again, the fine, partly burned coal, dropping down through the water-cooled firegrate, is burned on a subsidiary grate below.

At Cananea, where the first reverberatory was fired by powdered coal, the practice was given up in favor of oil, and so satisfactory has been the performance that a second furnace has been installed. One notes that when powdered coal was used the ashes from it accumulated in the outlet flues to the waste-heat boilers and there slagged, at the same time choking the flues. Using oil and allowing for the heat utilized for steam making, the net charge for smelting was 0.43 bbl. to 0.57 bbl. of oil per ton of charge smelted, and this too while melting a mixture of 37 per cent flue dust with 51.5 per cent calcined concentrates.

The Blast Furnace.

While the reverberatory has received a large measure of attention from metallurgists through the year, blast-furnace operation must not be forgotten.

Years ago W. L. Austin laid down the proposition that for real pyritic smelting 65 per cent of pyrite at least should be in the ore. Whether this figure is final or not, at least most of the sulphide ores treated need fluxing, and when this is true then coke must be employed to supplement the heat yielded by the sulphides themselves. A furnace in full operation accumulates hangings or accretions on the walls, but with jackets extending to the feed floor, or to the top surface of the smelting charge, such accretions are easily burned off. Formerly, when the walls were of brick, Austin proposed to prevent such hangings by "column charging"; that is, the charge close to the walls was to be of silicious or at least non-agglomerating ore, while within this and constituting the core of the charge were to be fed the sulphides so liable to sinter. I have tried it experimentally, but never have seen it adopted elsewhere.

There is less bosh in copper practice (7 in.) than in lead smelting, as the duty of the furnace in the first case is to do its most active work at the focus near the tuyeres and to complete the melting at the hearth. That this is by no means generally well done is evinced by the slagged tuyeres which must be continually poked to keep them open, and by the very considerable overfire as compared with silver-lead practice. Copper furnaces vary greatly in respect to overfire, and I have noted that when a self-fluxing ore is smelted there is less overfire than where much limestone flux is used. In spite of the endothermic effect of the limestone at the top of the charge, which effect continues up to a temperature of 800° C., the hot top prevails. Slag formation appears to be slow and the fire creeps up and keeps on top. With a fresh charge all looks well again, but before the next one is put in the gases have again risen to 700° C. to 800° C. and the furnace on top is a mass of seething flame, difficult even to look at. Meanwhile the blast is driven through narrow openings produced by the rodding of the tuyeres at high pressure and the hearth, except with excessive fuel, is too cold. We are not to think that this condition is universal. Lower blast pressures, good feeding and suitable slag composition assist in improving these adverse conditions.

Blast furnace jackets are now well built. The inside plate next the fire may be 5/8 in. for stiffness, the outside 3/8 in. Again there are no joints that can be attacked by the molten materials of the furnace, and jackets are made with welded joints especially at the tuyere opening.

The interior width of the blast furnace varies from 48 in. to 56 in., seldom more, and capacity is attained by adding to the length. Even at the Granby Works an addition of 3 ft. to the length is admittedly an advantage. Accretions are more easily removed and the active zone within the end jackets has been proportionately increased by the addition.

Increase in the height of the smelting column has resulted in the saving of fuel, but at the expense of the grade of matte, since the lower column with its greater overfire has caused the elimination of more sulphur.

At Cananea the crude ore charge carries 6.4 per cent Cu and hence, to get a good grade of matte, sulphur elimination is not so important as in the lower-grade ores. The matte fall is then 18 per cent and the matte carries 37.4 per cent Cu. I may add that they claim there is no trouble to keep the tuyeres bright and that the slag crust at the tuyere level is not hard. The tuyeres are barred or cleared out three times daily and are punched every three hours.

Converter Practice.

The great event of the year has been the development of basic copper converting. The process, using a Peirce & Smith basic-lined converter, was introduced at the Garfield plant of the American Smelters Securities Company, and of late these converters have been added to until there are six, five in continuous operation, the sixth as a reserve.

The converters are of the barrel type, 12 ft. diameter by 26 ft. long. The Peirce & Smith patents provide for the expansion of the basic brick-lined interior and for tuyeres that will not be broken by such expansion. The basic process itself is not patentable nor even the lining mixture used. In the United States basic-steel converting has been but little used, but its methods have been thoroughly developed in Europe and are a great aid to those who are now undertaking the conversion of copper matte.

A converter of the size above described will produce 40 tons of blister daily and will output 1800 tons to 4200 tons of blister copper on one lining with minor repairs at the tuyere region. As will be readily understood a low-grade copper matte presents no difficulty in treatment, so different from the case with the acid-lined converter. Within the year the Yampa smelter found it more profitable to sell matte to the Garfield works than to treat it locally. This matte was of too low grade to treat in the acid converter. Later on the Yampa plant was abandoned altogether and the mine output of 600 tons daily turned over to the Garfield works. When the latter plant lost the treatment of the Utah Consolidated ore they were glad to replace it with Yampa ore even at a low treatment rate.

To go back to basic converting: The Anaconda company has,

since last spring, been trying it out in ordinary converter shells with a high degree of success. They use as before the Repath independent tuyere and take up expansion by the use of wooden liners inserted in an occasional longitudinal joint. Around the tuyeres in place of the brick lining comes a packing of basic material with a suitable binding. In this way rapid work is done and the lining is comparatively stable and tight as against air leakage.

The apparent advantages of basic converting are many, especially for a new plant. Since the converter shells are not lifted by traveling cranes, and only such cranes are used as are necessary for handling the ladles, the superstructure can be made lighter, or much handling effected without the use of cranes. We thus do away with the detrimental effects of the sulphur fumes on the crane man.

While time is lost in operating a basic converter, such loss is less than with the acid-lined one.

Silicious ore can be freely used without needing clay or clayey slime as a binder, but at some plants such use is desirable because there is plenty of clayey slime made at the nearby concentrating plant.

The lining department and its machinery and upkeep, with the force needed for lining, is saved; there is but little drying out of linings required.

There are certain disadvantages in operating, but they are being overcome with greater knowledge.

In connection with the above, Arthur M. Day has developed a process, the outlines of which are not yet forthcoming, by which it has been possible at McGill, Nev., to produce slags running as low as 10.2 per cent SiO_2 and as high as 77 per cent FeO, or indeed any grade of slag desired. It can be readily seen that the quantity of slag is thus largely diminished, and in fact the whole process is hastened. On the other hand, in the acid-lined converter, the life of the silicious lining is correspondingly lengthened.

The Smelter Smoke Problem.

Copper smelting works have received severe lessons by disregarding the emission of smoke resulting from roasting and smelting the ores. Offensive as are sulphurous-acid fumes, the damage inflicted has been largely due to the presence of small quantities of sulphuric acid which has formed sulphates, to arsenic and particularly to solid particles of flue dust.

A variety of methods have been tried at great expense to overcome the difficulties, and the most successful has been the bag house proposed and tried 25 years ago for the saving of lead furnace fumes. When made of cotton the bags last but a short time if there is any SO_3 present in the gases, and even woolen bags have too short a life. The problem has been how to dispose of the comparatively small quantity of SO_3 to render it innocuous. This has been accomplished at the United States Smelting Company works in Utah by neutralizing the sulphuric acid by means of zinc oxide admitted to the flue where the escaping gases travel.

The Cottrell process of electrostatic precipitation of the dust is also making headway. In a small unit it has given satisfaction and now we learn that it has been worked up to a successful issue at the Coram plant of the Balaklala Copper Company. A single unit consists of a chamber 10 ft. x 10 ft. with a hopper bottom for the collection of the fallen dust. Within the chamber is a series of positively charged bars set between iron plates. These bars are made of two copper wires covered with strands of asbestos. The bars carry a current of high potential and give a positive charge to the dust particles which precipitates them on the low potential bars of iron. The plates are then jarred to shake the dust thus collected to the hopper bottom.

Salt Lake City, Utah.

The Camp Bird, Ltd., Ouray, Col., treated during the month of October, 1910, 6827 tons of ore, producing 7383 oz. of bullion and approximately 716 tons of concentrate. The statement for that month shows $109,000 applicable to profit.

A New Form of Electrode.

By A. Mahlke, Ph.D.

In this journal a number of articles has been published about the heat losses through the electrodes in electric furnaces. In the discussion of this object a point has been overlooked which might be of some importance in practice. The electrodes which were used in the investigations for determining the heat losses had a much smaller cross-section than the furnace. Consequently the current density in the furnace charge near the end of the electrodes (see Fig. 1) was much greater than in the interior of the furnace further away from the electrodes. The reason is, of course, that most of the lines of current start from the end of the electrodes, because the electrode resistance is much smaller than the resistance of the furnace charge and the current passes, therefore, through the electrodes as far as possible.

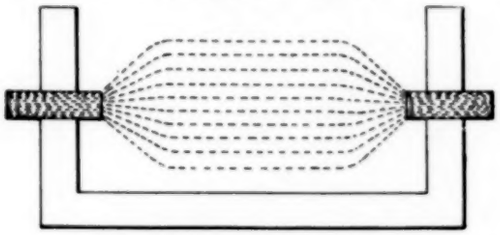


FIG. 1.—LINES OF CURRENT IN FURNACE.

On account of this greater current density in the charge near the end of the electrodes, the charge is at a much higher temperature near the ends of the electrodes than in any other part of the furnace, so that there is always a great difference in temperature within the furnace.

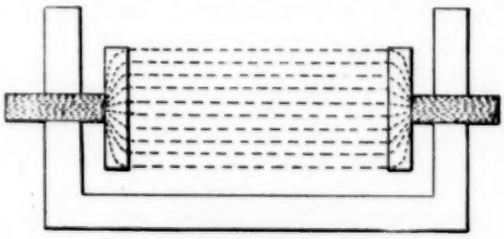
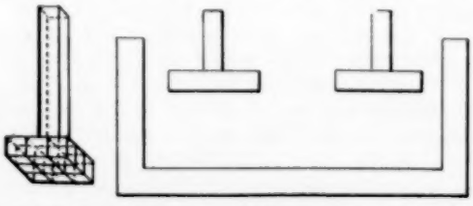


FIG. 2.—MODIFIED ARRANGEMENT.

This difference in temperature will exist during the whole process (though decreasing at the end of it), as long as the current in the furnace generates near the inner terminals of the electrodes much more heat than in the interior of the furnace, where the current density is less.

This difference in temperature also causes other inconveniences. The electrodes are heated to a much higher temperature than is necessary for the process in the center of the furnace and are burnt off to a greater degree and cause a greater heat loss than if the temperature of the whole interior of the furnace was uniform. To avoid these inconveniences one should fix at the end of the electrodes big plates of well-conducting carbon material, as shown in Fig. 2, which have the same cross-section as the furnace itself. These plates could be fixed to the electrodes by screws, wedges or any other contrivance made out of carbon, such as are used when short



FIGS. 3 AND 4.—ELECTRODE CONSTRUCTION AND USE IN ARC FURNACES.

electrodes are to be jointed in order to have electrodes of greater length.

By using such electrodes one would have the same current density throughout the whole furnace (see Fig. 2), and so the same amount of heat would be generated in each part if the resistance is the same everywhere in the furnace.

It would be practical to make the carbon plates thick enough to give them a considerable amount of heat capacity so that they can take a good deal of that heat generated in the part of the electrodes between the plates and the walls of the furnace, this part being heated to a higher temperature than the plates on account of the greater current density. But since some portion of this heat is conducted through the plates into the interior of the furnace, the heat loss is diminished as well as the burning off of the electrodes.

For resistance furnaces one could make such electrodes by packing carbon blocks around the end of the electrodes, as shown in Fig. 3. The resistance between the electrode itself and the blocks could be lowered by fixing the blocks to the electrodes with a graphite paste.

While these electrodes would be especially suitable for resistance furnaces, they might also be used for arc furnaces. The big mass of the plates forming that part of the electrode from which the arc arises (Fig. 4) would not get so hot as the thin end of the electrodes now generally used. So the amount of electrode material necessary for the process in the furnace might be decreased, which would be of importance for many purposes. The big plates only would be renewed, while the other parts of the electrodes carrying the plates could be used for many furnace runs.

Progress in Cyaniding During 1910.

BY R. W. GORDON.

So many specially trained engineers and chemists are now engaged on original work in connection with cyaniding that improvements are yearly increasing in number, if not in importance. The employment of educated men on mill work, too, has led to a more liberal policy in the publication of special information, both in the technical press and through constant interchange of visits and experience between cyanide operators. Progress, therefore, is proportionately more rapid as each man can commence his original investigations where his neighbor left off.

Nothing revolutionary has occurred in the cyanide field in 1910, improvements being chiefly confined to details, but the first mill using a continuous process throughout was placed in operation during the year, and the same mill adopted, with reported success, chemicals for aiding cyanide in the dissolution of values from a sulpho-telluride ore.

For fine crushing and grinding, especially on hard ore, stamps followed by tube mills have been specified in practically all the new mills, and much has been done in the past year, both in this country and South Africa, to properly apportion the work between them. Coarser and coarser material is being fed to the tube mill, and the point at which the cost of power and pebbles outweighs the initial expense and maintenance of stamps is a most important question, and one which must be solved for each installation.

If it were not for the nuisance of elevators and screens, rolls, crushing either wet or dry, would often replace stamps ahead of tube mills, and several plants using rolls and tube mills will be placed in operation early this coming year.

Chilian mills have held their own for crushing moderately soft material, and are attractive as occupying less space and costing less per ton for installation when foundations, bins, and buildings are considered. At the Goldfield Consolidated Mill a notable installation of these mills was made in 1908-9, where six Chilian mills were introduced between 100 stamps and six tube mills for intermediate grinding. At a comparatively low expense and without any addition whatever to the bins or building the capacity of the mill was materially increased. No data have been published to show whether any great economy has been effected by stage crushing and the elimination of the final undersize between each stage, but in a mill of this size stage crushing would certainly appear to be advantageous, for conditions could never be more ideal.

Considerable discussion is still taking place in the technical press between advocates of tubular and conical mills. So far no reliable data has been produced to show that conical mills are more efficient than the standard tubular type. The former seems to have given entire satisfaction in some places, and in others, again, particularly where a comparison has been possible, it has been discarded and replaced by the tubular. The use of steel rods instead of pebbles as a grinding medium in tube mills has been suggested and tried out on a small scale, but no millman has yet been induced to make a working test. Rather extravagant claims are also made that by the use of these rods material can be taken direct from the crusher and reduced to the necessary fineness at one operation, which is certainly a desideratum.

The use of shorter tube mills has been urged with the idea of removing as quickly as possible material which is already fine enough and returning the oversize to the feed end for retreatment. On the other hand, at El Oro a special attachment was recently added to the discharge end to prevent the oversize from leaving the mill. It would seem as if the accumulation of oversize at one end might have the effect of causing unequal liner wear.

It is now thoroughly understood that the efficiency of a tube mill depends almost entirely upon the consistency of the feed, excessive moisture being detrimental in every way. A dewaterer, therefore, as much as a classifier of coarse and fine material has become an essential part of tube-mill equipment. In individual cases, the Bayliss and Akin classifiers, which operate on the screw or worm principle, first used by George Moore at Mercur in 1903, have been installed for this work, and the Caldecott diaphragm cone is also used, but the Dorr classifier in its improved form was generally specified in 1910 installations, and appears to have given entire satisfaction.

In the year 1909 the continuous movement of pulp through a series of agitators was first adopted at the San Rafael mill in Mexico, after experiments on a working scale, in which both charge and continuous agitation were thoroughly tried out. At first thought it would appear possible for a particle to pass through three or four tanks without the necessary period of contact, but practically this chance is proved to be almost negligible and continuous agitation is now being used in many important plants, such as the Liberty Bell, in Colorado, and the Esperanza, in Mexico. By its means objection to the use of very deep air agitators has been overcome, as so little headroom is needed between the points of intake and discharge. Great care must be taken, however, to draw from one tank into the next a pulp having exactly the same consistency as that fed in, otherwise there will be an accumulation either of fine or coarse material. Such an adjustment can easily be made, however, and once made the agitators require practically no attention. This method has been applied successfully to so-called Pachuca tanks, to Hendryx agitators and to other forms of mechanical agitators.

With the objection to great depth removed, the Pachuca tank has become more than ever the favorite form of agitator, largely on account of the removal of all wearing parts from direct contact with the pulp. Among other agitators now in the market might be mentioned the "A Z," lately introduced from Australia, which employs practically the reverse of the Hendryx principle, and forces material from the center along the flat bottom of an ordinary shallow tank and up its sides. Although it consumes considerable power, it presents advantages over other forms, where heavy materials, such as sulphides, require agitation.

Continuous settling having been furnished by the Dorr thickener, and continuous agitation being also an accomplished fact, the need for a simple continuous method of separating valuable solution from valueless solids after agitation, or rather displacing the former by water, became more apparent.

Years ago, and before the invention of modern mechanical filters, an ingenious method of continuous dilution or decantation was evolved by John Randall in a small wet crushing mill near Deadwood. Intermittent decantation with its volumes of solution was then the order of the day, when Randall conceived the idea of reducing the amount of solution in circulation by using one solution only, and that a mixture of his final barren solution and water washes. This solution he continuously decanted off the final tank, used it for diluting the preceding one without return to the stock tank, again decanted it, and again used it for dilution until the first tank was reached. It thus gradually became enriched sufficiently to be worth precipitating, while the pulp moving in the opposite direction became impoverished. Randall's scheme worked perfectly on paper, but, unfortunately, the mechanical means then available for settling and pumping were unsatisfactory. It remained for Dorr to adapt the idea successfully to a series of his thickeners, first at the El Palmerito mine in Mexico and later in this country.

A simple continuous filter which will give perfect displacement is still badly needed, however, as continuous decantation yields tailings high in moisture and also, unfortunately, high in cyanide, especially if strong solution is in general circulation in the mill.

The Moore filter suit still drags along, and it is an open question whether Moore will eventually control the whole vacuum-filter field, as he undoubtedly deserves to, or whether it will be thrown wide open.

Merrill's pressure filter is being installed at the new Gertrudis mill, but it requires special conditions, whereas movable or stationary vacuum filters are applicable everywhere and give uniformly good results. Various forms of continuous revoluble filters have been placed in operation during the year, but so far they are too complicated to be widely adopted, and none of them compare with the intermittent vacuum filters in the displacement of valuable solution. South Africa has at last followed this country in adopting vacuum filtration instead of intermittent decantation, and we presume will shortly be trying out continuous agitation and decantation also.

Separate treatment of sand and slime is seldom considered in a modern mill, but mention must be made of a novel method in use at the new Portland mill, in Colorado. There separate treatment is used, but the commercial values in the sand are in solution when the classifier is reached, and all that is needed is a wash.. The thickened sand from the classifier is, therefore, led through a second classifier—or, more properly, a dewaterer—containing barren solution, and from there, through a third containing water, to the dump.

The possibilities of the Clancy chemical process have become more and more apparent, and the reported success of the Crow-Tippet method at the Portland mill has but whetted the appetites of producers of sulpho-telluride ore for a mill operated under straight Clancy principles. Such a mill will shortly be built in the Cripple Creek district, and a satisfactory economical outcome would appear highly probable since the Clancy claims have been published. The field for such a process is large and profitable, as quantities of ore are known to exist containing values in sulphide form, but are too low grade to afford roasting. The operation of the first plant will, therefore, be watched with much interest.

For precipitation in the larger mills the use of zinc shavings seems to have given place almost entirely to zinc dust, the press generally installed being the Merrill triangular press, which was perfected at the Homestake. One great argument in favor of the use of a zinc dust press is safety of the precipitate, either in case of fire or of light-fingered employees.

On the whole, the year 1910 was well spent by the cyanide fraternity. Costs were reduced, tonnage increased and more bullion produced by the cyanide process than ever before.

A Vertical Magnifier

By H. M. Boylston.

The little instrument[1] illustrated in Fig. 1 was devised in this laboratory as the result of a long search for a simple, convenient and inexpensive means of measuring the impressions made in metal specimens by the Brinell ball test for hardness.

It consists of a doublet lens made up of two plano-convex lenses having a combined focus of 38 mm. and adjusted in a cylindrical mount about 3 in. high and open at the bottom, as shown in the figure. Provision is made for vertical illumination and a small scale is added for measuring small distances accurately. The focusing glass used by metallographists makes a satisfactory basis for constructing the instrument, the lenses in this being specially corrected for flatness of field and sharpness of focus.

A brilliant illumination is obtained by inserting a plain glass reflector *G* in a slot cut in the cylindrical mount at an angle of 45°, one side of the cylinder being left open to admit light from any convenient source (daylight, incandescent electric lamp, Welsbach gas burner, etc.). The path of the light from its source to the specimen to be measured or examined is indicated by arrows in the illustration.

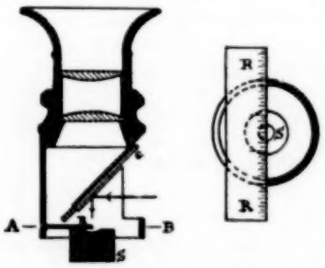


FIG. 1.—VERTICAL MAGNIFIER.

A thin steel scale *R* with proper graduations on one edge is inserted near the bottom of the cylinder in a slot cut for that purpose and is adjusted so that the graduated edge corresponds to a diameter of the cylinder.

The instrument is held in one hand with the opening directed toward the source of light, while the specimen *S*, with impression uppermost, is held against the steel scale with the other hand in such a position that the scale corresponds with a diameter of the circular edge of the impression, one of the graduations of the scale being set tangent to an edge of the impression. The diameter of the impression is then easily read off on the scale which with the impression itself has been magnified considerably by the lenses of the instrument. A scale graduated to 0.2 mm and hence easily read by interpolation to 0.1 mm has been found very satisfactory, since no greater accuracy than this is warranted; for the impressions obtained in this test often are not perfectly circular and the lip of the impression is often rounded instead of being sharp. Steel scales graduated to 0.1 mm can be obtained, however.

Although designed primarily for the purpose just described this instrument has many other uses. It can be used to great advantage in examining fractured surfaces, or for examining the polished surfaces of metallographic specimens for flaws in the polishing, or for minute blowholes. It will also lend itself readily to any case where a low power magnifier is needed or where small distances are to be measured rapidly with the degree of accuracy already indicated. It is hoped that the magnifier will be found useful in many cases, in the machine shop, for instance.

The focus of the instrument may be adjusted within limits for large specimens, in which case the scale is removed from the slot and laid directly on the object to be measured, the magnifier being placed above and against the scale.

Metallographic Laboratory of Harvard University.

[1] Another somewhat less convenient, but fairly satisfactory device of this sort is described on p. 53, "Laboratory Experiments in Metallurgy" by Sauveur and Boylston.

Metallurgical Progress in Utah and Nevada During 1910

BY ROBERT H. BRADFORD

Salt Lake City is conceded to be the most important smelting center in the United States. The tremendous ore supply of Bingham, Park City and Tintic gives a combination of easily smelted mixtures and has justified the building of exceedingly large smelting plants near Salt Lake City. The favorable position of this region as a railroad center enables the smelters to draw large supplies of ore from all parts of Utah, from Nevada, Idaho and even from California. The ore supply is more diversified than anywhere else in the Union.

Large custom smelters with their keen competition for custom work have given smelting rates more favorable to the seller than those obtaining elsewhere.

The large companies maintaining custom plants in this region include the American Smelting & Refining Company, the United States Smelting, Refining & Mining Company, the International Smelting & Refining Company.

The Murray plant of the American Smelting & Refining Company has been operating during the year at somewhat reduced capacity as has been common to most of the lead plants of this company. The improvements in the pot roasting process have reduced the cost of roasting the sulphide ores and have given very desirable materials for blast furnace smelting. Record runs for lead blast furnaces of dimensions 46 in. x 168 in. have been made during 1910. The greatest tonnage for a single day was 318 tons of ore and flux, exclusive of fuel and returned slag. Average daily tonnage for one furnace for a whole month was 280 tons. These figures show the highest tonnage yet reported from any lead smelting plant.

The Garfield plant, a copper smelter, has operated on increasing ore supplies during the year. The concentrates from the gigantic concentrating mills of the Utah Copper Company, from the new mill of the Ohio Copper Company, along with ores and concentrates from numerous mines of the surrounding country have taxed the reverberatory and blast furnaces to their limits.

The basic lined converters of the Peirce & Smith type have done excellent service. By means of the process using basic lined converters the cost of converting at Garfield has been notably lowered. The converter shell is 23 ft. long by 10 ft. in diameter. Next to the shell, along the bottom, is a lining of ordinary fire brick varying in thickness from 4 in. to 13½ in. This lining is thickest along the bottom and becomes thinner as it rises to the sides. Inside this lining is the basic lining of magnesite bricks. The circular throat is at one end. Matte and silicious ore are charged through the throat. Thirty-two cast-iron tuyeres are provided, each with special valve and flexible hose connection to the air main. An opening called the breast opposite the tuyeres provides for the removal of the slag and metal.

The initial charge of matte consisting of four or five ladles of matte (40 tons) is poured in through the throat and about three tons of dry silicious ore are added. The blast is introduced and the converter rotated until the tuyeres are well below the surface of the charge. When the blow is well begun the blast pressure reaches 10 lb. to 12 lb. per square inch. When the silicious ore has been fluxed the slag is poured and additional amounts of matte and ore introduced. The operation is repeated and continued until enough "white metal" is formed, when the addition of matte and ore ceases and the white metal is blown to blister copper.

Care is taken to thoroughly dry the silicious ore before charging it into the molten matte to avoid explosions.

At the United States lead smelter at Midvale the arsenic plant is of interest. The blast furnace flue dust collected at the bag house is returned to two Brinton roasters having rotating hearth and stationary rabbles. Each furnace has capacity of 10 tons per day. The dust heated to redness in this furnace is in large part volatilized. The residue from the hearth goes back to blast furnace. The fumes from the Brinton furnace consist principally of arsenious oxide (As_2O_3). The sublimed arsenic product is collected in zigzag flues built of brick masonry. The product from these flues is taken to a reverberatory furnace and revolatilized or resublimed, the fumes again collected in zigzag flues. The product from the second set of flues is practically pure As_2O_3. It is taken from the flues in cars and transported by a monorail tram to the packing room, where it is packed in barrels with capacity of 400 lb.

Many improvements in converter roasting are being perfected at the United States plant. Much flue dust is being sintered into a form suitable for blast furnace treatment. None of the ores fed into the rectangular converters receives any previous roast. Thin layers of ore and short time for roasting a charge have increased the capacity of the roasting plant. Careful mixing and moistening of the ingredients have assisted materially in the converter roasting.

The International Smelter at Tooele, Utah, was put into commission on July 25, 1910. This plant is entirely new and thoroughly up to date. Construction work was commenced Dec. 9, 1908, just 19½ months before the first furnace was blown in. Considering its extent and the massive construction this plant holds the record for speed of construction.

The smelter was built primarily to treat the ores from the Utah Consolidated Mining Company's mines at Bingham and the South Utah mines at Newhouse, Utah, but is provided with capacity to treat also much custom ore. The Highland Bay smelter at Murray originally treated the Utah Consolidated ores, but this plant was closed down as the result of injunction proceedings brought by the farmers of Salt Lake County.

The ore is sent from Bingham to the International by a Bleichert aerial tramway four miles long built over a very rugged route. The tramway has a capacity of 125 tons per hour. The receiving bins at the smelter are of steel with a capacity of 10,000 tons. The ore from these bins is conveyed to the sampling mill by belt conveyors, where it is crushed and sampled. From the sampler the ore is conveyed by belt conveyors to the receiving bins at the McDougall roasters. From these bins the ore is again conveyed by a belt conveyor to the charge hoppers of the McDougall roasters. In transit from the bins to the roaster plant the ore is weighed automatically as it passes over a Blake-Denison continuous weighing machine.

The roaster plant contains 16 McDougall furnaces of 45 tons daily capacity. From these furnaces the ore drops into hoppers arranged directly over the tracks of the electric tramming system for transportation to the hoppers of the reverberatory furnaces.

Five coal-fired reverberatory furnaces 19 ft. x 102 ft., with a grate area of 7 ft. x 16 ft., are provided, each with a capacity of 300 tons in 24 hours on natural draft. The flame from each furnace passes through a 746-hp Stirling boiler and develops 600 hp of energy. The slag is skimmed into slag cars of 225 cu. ft. capacity and hauled over the electric tramming system to the slag dump. The matte tapped from the side of the furnace runs through launders directly into the converters.

The converters are of the barrel type 96 in. x 150 in. and are operated electrically. A 60-ton electric traveling crane is provided to serve the converters. The casting room has a 30-ton crane and there are two small 12½-ton cranes used to transfer the slag from the converters directly to the reverberatory furnaces in the molten condition.

The metallic copper is poured into the casting ladle and cast into bars ready for shipment to the refinery.

The following notes were made and published* by the writer on the occasion of the visit of the Utah Society of Engineers to the International smelter:

*Salt Lake Min. Rev., Nov. 15, 1910.

A striking feature of the new International smelter as observed on the trip was the massive steel construction everywhere apparent giving large roomy buildings with high roofs affording excellent ventilation. Surely the idea of supplying the workmen with good air is worked out successfully.

The means of transporting materials about the plant was a most interesting feature. The standard broad gage tracks for the electric locomotives allowing for the movement of large railroad cars anywhere about the works, the free use of the belt conveyor with its automatic tripping and weighing devices, the unusual adaptation of the electric traveling cranes for ladles and converters provide an ease and economy of movement unsurpassed in modern smelting practice.

The careful sampling along with the exact weighing of materials makes it possible to check one department against another and thus place the responsibility for any losses due to careless operating at any stage of the smelting process.

The plant has an excellent location, being near the mines supplying most of its ore, the mountain range intervening being of little consequence, thanks to the aerial tramway; also near two through railroad lines and yet away from the agricultural districts. The gently sloping site, allowing for extensive building construction on any one level, and yet with enough relief for the materials of the smelting operations to be moved horizontally with but one lift, combines the good features of the terraced and the level site. The extent of the grounds laterally will allow of the addition of extra units with ease.

The simplicity of the process by which they pass from the mine ore to copper bullion illustrates the tendency of most recent commercial practice, but three operations being required, namely, roasting, smelting and converting. Contrasting this with the Welsh method in operation in Wales when that country was looked to as the leader in copper smelting, we see the operations cut to one-third. This Welsh method involved nine different operations, consisting of partial roasting, smelting, another roasting, a second smelting, etc., each process consuming fuel, material and energy far out of proportion to the advance of the resulting product toward the copper bullion desired. The "conservation of resources" is everywhere apparent at the International. The sulphur of the ore in the McDougalls furnishes all the heat for roasting and part of that needed in smelting. The roasted ore is not cooled, but charged hot into the reverberatories where it receives the additional heat necessary for fusion. The fuel of the reverberatories gives heat for the fusion and for the generation of steam more than sufficient for all power about the works. The matte in the converters is kept sufficiently heated by the burning of its iron and sulphur by the air forced into it so that the metallic copper is taken out well above the fusion point, allowing it to be poured slowly into the molds ready for market.

Cyaniding.

Utah enjoys the distinction of having the oldest cyanide plant in the United States. The Mercur mill was the first to try out the McArthur-Forrest process in this country. The Mercur mines are still producing handsomely and the ore, while not treated now in the original mill, is still responding to the cyanide process in the improved De La Mar mill now owned by the Consolidated Mercur Gold Mining Company.

Until the beginning of the present year the roasted ore was not classified and treated in the filter plant, but was kept separate from the oxidized ore. Now the roasted ore is mixed with the oxidized before the cyanide solution is added and all ore is classified and either leached or filtered according to its fineness.

An increased extraction is claimed from passing air up through the leaching pipes into the sands at intervals during the leaching. Experiments are being carried out with a view to using zinc shavings instead of zinc dust for the precipitation of the gold.

The company is now retreating old tailings to the extent of 100 tons or so per day. The supply of base ore from the mine is amply sufficient to keep the mill supplied with the 700 tons per day required.

The old tailings dump at Manning, four miles from Mercur, is being treated by the Manning Leasing Company. These tailings from the old Mercur mill carry $2 to $2.50 gold per ton. About $1 of this value remains in the tailings after present treatment. The company is using the Holderman filter process very successfully. It treats 160 tons of tailings per day at a cost of 75 cents per ton. The tailings from the old dump are conveyed to bins and sluiced with cyanide solution from bins into Chilian mills. All material passes through 18-mesh screen and is run by gravity to Holderman tanks fitted with filter leaves. The solution runs by gravity through canvas leaves and then to the gold tanks. Wash water or weak solution is run into tanks for 16 hours and the pulp is then sluiced over the dump. All clear solution from filter leaves goes to gold tanks and again through Holderman leaves to zinc boxes. The gravity filter leaf in the gold tank does excellent service, as is evidenced by the crystal clear solution as it enters the zinc box.

The Daisy Combination mill at West Dip, five miles west of Mercur, is milling 140 tons of ore from the company's mine. The process employed here is much the same as that used by the Consolidated Mercur mill.

The Utah Leasing Company, operating the Old Telegraph Mine at Bingham, has built a 50-ton cyanide mill to treat the silicious ore that the former company had left in the mine owing to high penalties for silica imposed by lead smelters. The ore is first crushed by a Ferraris-Blake crusher to ½ in. in diameter, then fed by an automatic plunger feeder into a tube mill. Cyanide solution is added as ore enters the tube mill. From this mill the pulp, after addition of more solution, is classified. Sands return to the tube mill. All slimes are agitated in Pachuca tanks for 36 hours. Pulp is then transferred to 100-ton settler. Thickened pulp goes to Kelly press and all clear solution to zinc boxes. The cost is $1.50 per ton for an extraction of 82 per cent.

Nevada.

During the years 1906 and 1907 the Western mining States enjoyed a period of prosperity previously unheard of. No one camp or no two or three camps were responsible. The entire mining West shared in the prosperity. If any State received more attention than another Nevada was that State. Her mountains were hurriedly prospected, with most flattering results. Many rich gold camps sprang up and the gold and silver ore taken from the surface outcrops created such a stir that it became common understanding that a gold prospect in Nevada was more attractive than such a prospect anywhere else. In fact it was not uncommon for the prospector to succeed in selling his claims simply because he could show they lay in Nevada. Purchasers seemed willing to buy geography.

It is not strange that many "wild cat" schemes were floated and that many investors lost their money. Nevertheless the fact remains that the gold and silver properties of real value opened up at that time and later show a most wonderful richness, and Nevada is now and will continue to be one of the foremost States in the production of the precious metals.

During the past year the State has furnished the mints with gold bullion in advance of any previous year. The mines in all parts of the State have added their quota, making a total that is enviable. While the large metallurgical plants like those of the Goldfield Consolidated, the Tonopah Mining, the Montana-Tonopah, etc., have operated regularly with large monthly shipments of gold and silver bars, the metallurgical feature of striking importance is the erection of the many smaller mills throughout the entire State. In many cases transportation facilities have not warranted the shipping of ore to custom works at any considerable distance and individual mines have built their own mills. Where the production of the

small mine would not justify the building of a mill, a number of small producers have contracted a certain small tonnage each to a custom mill to be installed near at hand. Thus 10-stamp mills, 20-stamp mills, 30-stamp mills have been put into commission or contracted for very widely.

The metallurgical progress of recent years has made this possible. The recent development in cyaniding silver minerals, the improved methods of getting the gold ore finely crushed and cheaply filtered have assisted in making possible the treatment on a small scale of the gold and silver ores of the State.

It will be impossible to give even a brief description of the mills put into commission and those being erected during 1910, and I shall attempt only to give some data regarding a few of them. In many instances the gold is coarse enough to be readily caught by the amalgam of the plates. Where the ore is in condition for a good extraction to be obtained by amalgamation alone, stamp milling with apron plates is all that is employed. In other cases the coarse gold only is amalgamated and the fine cyanided. Where the gold is partly in combination with sulphides present, concentrating machinery is also employed and two marketable products are obtained, bullion and concentrates. The concentrates are often shipped to the smelters, but may be profitably treated by special process at the mill. Cyaniding after fine crushing with long time and strong solution is effective in extracting the gold from the concentrates, and this process is resorted to at some of the mills. Then chlorination after roasting is employed with the concentrates at the Goldfield chlorination mill with excellent extraction.

The 60-stamp mill of the Tonopah-Belmont Company at Millers has operated upon 220 to 250 tons of ore per day.

The Tonopah-Liberty Mining Company has put into commission its mill at Liberty near Tonopah. The power for this plant is obtained from the Nevada-California Power Company. The all sliming process is employed and the mill is a model of modern construction. The ore passes through a Blake crusher, Chilian mill and tube to be reduced to slime, then a Trent mechanical agitator, combining power and air agitation, and finally to the Oliver filter. The mill has been in commission since Oct. 2 and is treating 130 tons per day.

The 30-stamp mill of the Tonopah Extension Company is treating 135 tons per day, obtaining concentrates to the amount of 40 tons and bullion valued at $45,000 each month.

The Bonnie Clare mill in the Gold Mountain district, Nye County, started up in September prepared to treat the company's ore as well as custom ore, stamp milling, amalgamation, concentration and cyaniding being employed.

The 10-stamp mill of the Nevada Reduction Company at Manhattan is a good type of the small mills recently erected and now building. After the stamps come the silver-coated copper amalgam plates, then the classifiers (in this mill the Dorr is used) giving sands for tube mill and slimes to join the product from the tube mill going to the filter (the Oliver is used here). The gold is precipitated by zinc shavings in the zinc box,

Small 10-stamp mills or mills of this capacity are to be found in operation or building at Rhyolite by the Tramp Consolidated; at Dayton, Lyon County; at Blackhorse by the Amalgamated Nevada Company; near Manhattan by the Manhattan Milling & Leasing Company; near Caliente by the Gold Chief Mining Company; in Esmeralda County by the Gold Bar Company, and at various other localities.

Larger mills ranging from 50 tons to 150 tons capacity are operating or building by a number of other companies, including the Nevada Hills Company, at Fairview, Churchill County; the Rich Hill Company, at Eldorado; the Rawhide Queen and Rawhide Coalition—the Queen mill is now running; the Austin-Manhattan Company, with its dry concentration process with the Huff electrostatic machine; the Rainbow-Amazon Company, at Jarbridge, Elks County, etc.

The Butters cyanide plant in Six-Mile Canyon, Storey County, is closed for the winter. During this time it will be put into first-class condition.

The Bamberger-Delmar 500-ton cyanide plant has been sold to the Great Western Machinery Company, of Denver, who are dismantling and disposing of the plant. The above are some of the mills that have been installed or undergone changes during the year, but it by no means embraces all that might be mentioned.

The Mason Valley Mines Company is erecting a smelter at Wabuska to treat the ores from the Nevada-Douglas and many other Nevada mines.

The mammoth Steptoe smelter at McGill, Nev., has been in full operation and has reached an output of 6,000,000 lb. of copper per month. The cost of smelting is kept down below that of most modern plants. The future of this plant along with the concentration mill of the company is most promising. The shipment of an entire trainload of copper bullion—25 cars of 50 tons each—from the Steptoe smelter to the Eastern refinery will impress one with the magnitude of the company's operations.

Gold and Silver.

Bullion Refining.—The details of electrolytic refining of cyanide precipitates at the Waihi Gold Mining Company, Waihi, New Zealand, are given in the *Aust. Min. Stand.* for Oct. 26, 1910, by J. G. GOOSMAN, who has been identified with the work in every department.

The zinc boxes are cleaned weekly at each mill, where the gold-silver alloy is prepared for subsequent treatment at the electrolytic refinery. The zinc slime is treated with 10 per cent sulphuric acid in wooden vats 9 ft. in diameter and 6 ft. deep. Agitation for one hour usually dissolves the zinc, after which the slime is washed with hot water, the solution decanted and the slime filtered by vacuum pump. The dry slime is then mixed with flux, consisting of 30 per cent of its weight of borax, tamped into iron trays and dried in an oven.

The custom of melting this material in plumbago pots, followed by cupellation of the metal, was abandoned some time ago, but will be adopted again on account of the inherent difficulties in the present system. At present the melting and cupelling are both carried on in a "test," but a great deal of trouble is experienced in getting mixtures which will withstand the action of the borate slag, and that of the litharge and copper oxide. The "tests" now used have a foundation of cemented firebrick and sides of cement 3 in. thick. The absorbent body is well tamped in and allowed to stand several weeks before using.

For use the "tests" are heated in a furnace under good blast and the charge is added with scoops. As soon as molten the "test" is tipped to allow the slag to run off. When all the slag has been run off the first lead is added, which amounts to 10 per cent of the weight of the dry slime. The cupellation is hastened by a blast of air through a tuyère, which drives the litharge over the breast of the "test." When the first litharge is almost removed, the retorted gold from amalgam plates is added, and a second charge of lead, 5 per cent by weight of the slime, making in all 15 per cent of lead added in cupellation. In the other system of melting in pots followed by cupellation only one charge of lead is used, amounting to 10 per cent of the weight of the bar bullion.

The slag from the melting operation and the litharge from the cupellation are reduced in a small blast furnace, and the resulting lead is either refined if rich in bullion or liquated and used again in cupellation. The gold-silver alloy is ladled from the "test" into molds 10 in. x 8 in. x ⅜ in., each bar weighing about 200 oz.

The electroylic plant has 30 cells, each having a capacity of 10 gal. of parting solution, 0.6 per cent nitric acid. In each cell are filter frames holding four of the bullion bars, with the anode terminal on the adjoining edges of the bars. The 4 sq. ft. of cathode surface is a ¼-in. silver slab fitting the bot-

tom of the bath and having a 4-in. silver ribbon connection with the cathode terminal. The current used is 100 amp, or a current density of 25 amp per square foot of cathode surface. The voltage averages 95 volts, or 3.16 per cell. This figure varies with the temperature of the bath and with the decreasing distance between anode and cathode. The precipitated silver is scraped from under the filter cloths with wooden scrapers, washed to remove acid, melted in pots and cast in bars of 1000 oz., 998 fine. The gold and bullion scraps from the filters are washed through a 10-mesh sieve, the fine gold washed free from nitric acid and boiled for three hours with concentrated sulphuric acid. After another washing the gold is melted in pots and cast into bars of 1000 oz., 997 fine. The silver from the acid solutions is precipitated with salt and reduced with zinc, sulphuric acid being added to hasten the action.

Copper.

The Converter Flame.—The process of copper converting is indicated and controlled largely by the appearance of the flame issuing from the nose of the converter during the blow. The changes occurring are not as easily described as observed. An interesting article dealing with the successive stages of converting as indicated by the converter flame has been contributed to the Institution of Mining and Metallurgy by Mr. Donald M. Levy. The article is published in *Bulletin* No. 74 and is accompanied with reproductions of color photographs taken on Lumière plates, the exposures being made at night in the converter-room at the Washoe Smelter, Anaconda, Mont. The illustrations are excellent reproductions of the various flames, of which four are shown in the original paper.

The first shows the condition at the beginning of the blow, and is a rich red-brown flame due to the coal added, to the burning of iron and sulphur, and partly to the zinc, lead and arsenic in the fume. The flame continues red to red-purple for from two to eight minutes, after which the green flame appears and the slagging stage has properly begun. The green becomes more and more prominent, and after about 15 minutes is a vivid apple-green. This is shown in the second color plate, and is typical of the slagging stage.

The white-metal stage is indicated in the third color plate, where the green has disappeared and flashes of blue appear, resulting in a white-blue flame. At this stage the slag is poured off, and if the converter is too hot it is "doped" or charged with scrap copper.

The second part of the blow, from white metal to blistered copper, commences with a red-brown flame and smoke, but finally thins to a red-purple flame and continues thus to the end of the operation. This is shown in the fourth color plate. The end of the blow is difficult to judge, but is determined by the emission of small shots of hot copper, which no longer adhere to the hood over the converter, but rebound from it.

The green effect during the slagging stage is due to the formation of iron silicate, while the blue flame at the end of the slagging stage indicates that the iron has been slagged and that copper silicate is being formed. These colors are not produced unless the respective oxides are being slagged.

Following are observations made during a complete blow:

Converter A.—Matte charge.

Charged 2:25 p. m.—Flame red.
2:29 p. m.—Green appears.
3:00 p. m.—Green becomes brighter.
3:17 p. m.—Vivid green.
3:18 p. m.—Skimmed; no metal reached.
3:21 p. m.—Blue appears in flame.
3:27 p. m.—Flame chiefly blue; converter turned down to skim. Doped.

Converter B.—White-metal charge.

Charged 4:13 p. m.—Doped.
4:14 p. m.—Converter turned up; flame red.
4:16 p. m.—Flame red-purple.
4:18-5:20 p. m.—Flame peach to bronze-purple color.

Ore Dressing.

Classification.—Metallurgists generally recognize the great service which Prof. Robert H. Richards has rendered to the industry of ore dressing. They will be glad to study his latest contribution on the development of hindered settling apparatus, which has been published in the November and December, 1910, issues of the *Western Chemist and Metallurgist.*

The causes of loss in concentration are first considered, and then methods of voiding the losses. The author find four ways in which mineral loss may occur: as including grains, as long or flat grains, as fine free mineral and as absolute slime. The treatment of flat, long or included grains will involve recrushing to a size which liberates the maximum quantity of mineral while making the least slime. Fine, free mineral in jig, table and vanner tailings offers the ripest field for improvement in milling, and the object of the present paper is to consider methods for saving this product. In regard to absolute slime, the author is still in doubt whether they can be saved by any means or not.

Free Settling.—The author then takes up four methods of making free settling tests to determine the ratios of quartz and galena which are equal settling. The methods are: (1) by letting clusters of grains settle together; (2) by letting individual grains settle one at a time; (3) by spitzlutte and microscope to get the ratio, and (4) by spitzlutte and photograph to get the ratio. The four methods give nearly the same results, viz., 3 to 3.5.

In the free-settling classifier the following things are essential: (1) means of feeding the pocket with the mixed sand to be treated; (2) rough sorting in the pocket, raising the finished light material by the rising water from the sorting column, and settling the heavy unfinished product to the sorting column; (3) finishing the sorting in the sorting column, allowing the heavy finished grains to settle down and out while the entangled grains are raised to the pocket again; (4) finally, admission of wash water into a pressure box, where the stream is divided, part rising and doing the sorting, the remainder passing out of the spigot with the discharged sand. The third point is most difficult to obtain, as there must be no downward currents to carry slime into the spigot, and the sorting column must have the same rising velocity in all parts of its horizontal cross-section. Classifiers are good or bad according as they do this one thing well or poorly.

What happens in a free-settling classifier is that, with the exception of the first spigot and last overflow, all the products are truly classified, i.e., they contain larger grains of quartz and smaller grains of mineral. The first spigot has an added increment of coarse grains of heavy minerals, and the last overflow has an added increment of fine material in addition to the truly classified grains. Hence, these products require special consideration in concentration.

Hindered Settling.—Under this head the author presents some new figures for the hindered settling ratios as determined in a new series of experiments. The earlier ratio of quartz and galena under hindered settling conditions was 5.85, but in the new series of figures it is 6.92. This figure is true for either continuous or pulsating current, but the latter has an advantage in avoiding the harmful downward currents. Hindered settling thus gives a wider range between the particles of quartz and galena, which are equal settling, than is possible with free settling, the ratios being about 3.5 for free settling and 6.9 for hindered.

The essential feature of getting hindered settling is a hindered settling chamber above with a constriction below, and the whole current of water rising through the two. The cause of the difference between hindered and free settling seems to be that with the latter both quartz and galena are settling in water with a specific gravity of 1, while with the former the grains are settling in a denser medium, probably specific gravity 1.5, due to the crowding of the mineral grains. The effect of hindered settling is twofold: (1) The slime is more per-

fectly kept out of the spigot product and (2) the advanced ratio helps the concentrating machines more than free settling.

The requirement for hindered settling is a proper ratio of area between the settling chamber above and the constriction below. The author has determined this to be 1 to 4, and believes that this will effect a condition of "full teeter" of the bed in the teeter or settling chamber and a discharge through the constriction below.

The author then takes up a consideration of jigs and shows that in the ordinary jig the act of pulsion is identical with that of hindered settling current. The return stroke, or suction, causes an intermission in this action, so that the grains of galena cannot continuously settle through the quartz under hindered settling conditions. In the pulsator jig, however, the conditions of hindered settling are maintained continuously and to this is ascribed the remarkably high capacity of this machine.

Five types of hindered settling classifiers are described. Just as the free settling classifier has an added increment in the first spigot and last overflow, so has the hindered settling classifier, with this distinction: In the latter the larger ratio between the diameters of the grains in juxtaposition is greater, and as a consequence the coarsest product has a larger proportion of all the values in it, and the water in the spigots and overflow are reduced. Fine free mineral grains likely to contaminate the tailings of the succeeding concentrating machines are more easily kept out of those tailings.

Finally, the author takes up various concentrating machines, such as the jig and table, explaining their faults and showing the benefits to be derived from classification or screening.

The original article is long, but will well repay study. Some of the data presented have been published, and numerous references are made to former papers by the author. In addition there is a great deal of hitherto unpublished work, illustrated with numerous cuts.

Ore Sampling.

The Philosophy of Sampling.—The correct sampling of ore is a necessary preliminary to good metallurgical work. In the *Canadian Mining Journal* for Dec. 1, 1910, JOSEPH T. MANDY presents some observations on this subject. Mechanical and hand sampling have had their supporters, and each side undoubtedly has some strong arguments to present, but the weight of argument favors the former method.

The author inclines to the belief that the whole matter of sampling resolves itself into a question of averages, and that in the long run the chance of error favoring either buyer or seller is rather remote. But since the accuracy obtainable varies inversely as the chances for error, he believes that such chances should be eliminated to the utmost. With each chance for error eliminated, the sampling becomes more accurate.

Mechanical Sampling.—The initial feed is of the greatest importance. It should be regular, and hence the author prefers hopper feed to hand-shovel feed. With the latter method the feed is like to be intermittent, and consequently the first cut of the sample may be irregular and incorrect. If hand-feed is adopted it would be necessary to have some intermittent hopper between the crushing apparatus and sampling machine. The same objection to intermittent feed applies to the subsequent treatment of the first cut, for no matter how continuous the original feed may be, it becomes an intermittent one after the first cut unless some device is installed to prevent it. The installation of rolls partly removes this objection, but not wholly, and it becomes apparent that some regulating device should be used after each cut of the sample. Another objection to mechanical sampling is that in the swing of the cutting device coarse particles of ore strike the edges and are likely to be thrown into the wrong compartment. In the ordinary Vezin machine the particles will always be thrown into the reject compartment, but with the Brunton stream diverter it is less likely to occur, while with the Brunton oscillating cutter the possibility of error has an opportunity to average. The greatest point in favor of the mechanical sampler is the elimination of the personal element, which is ever present in hand sampling.

Hand Sampling.—Here the personal equation is the greatest drawback, and no matter how conscientious the operator, his precautions against unfair or incorrect work are insufficient. This becomes true even with unskilled labor, for such operators soon learn the points of the process and eventually work incorrectly. Fractional shoveling, or taking every fifth or tenth shovelful of the ore, is a very poor method save for the poorest ores. Coning and quartering is an operation that is open to the strongest criticism on account of the uneven distribution of the coarse and fine ore in the cone and cake. This method of finally reducing the sample to small size is condemned by the author regardless of whether it is used in connection with preliminary mechanical sampling or hand sampling. A tabulation of figures is given showing the segregation of sizes of ore in a cone and in the resulting cake from which the quarters are rejected or saved.

The author concludes that in all fairness it should be acknowledged that mechanical sampling is superior to hand sampling, but that the latter is open to improvement in the points mentioned. In addition, it is urged that the cutting of the sample should be made mechanical clear through the operation, substituting riffles for the coning and quartering system.

New York Meeting of American Institute of Chemical Engineers.

The third annual meeting of the American Institute of Chemical Engineers was held at New York City from Dec. 7 to 10. The sessions were held alternately at the Hotel Astor, Columbia University and the Chemists' Club. The excursions, a full program of which was given in out last issue, had been carefully arranged so as to be of as manifold and actual interest as possible and were greatly enjoyed; the chain of excursions, distributed over several days, has become one of the most prominent features of the Institute meetings.

Dr. Charles F. McKenna's presidential address dealt in a very interesting way with the Evolution of Portland Cement Processes. We hope to publish the address in full in one of our next issues. An abstract of it and of the other papers presented will be found in the synopsis in this issue.

An especially interesting feature of the proceedings was the discussion on chemical engineering education. Dr. F. W. Frerichs presented the report of the committee on chemical engineering education. Dr. Fred W. Atkinson, president of the Polytechnic Institute of Brooklyn, presented a paper on the development of the chemist as an engineer. Prof. M. C. Whitaker, of Columbia University, presented a paper on the training of chemical engineers which meets the requirements of manufacturers. The general discussion of the subject extended over several sessions. Prof. Whitaker's paper is printed in full elsewhere in this issue.

The election of new officers of the Institute for the coming year gave the following results.

President—Dr. F. W. Frerichs, Herf & Frerichs Chemical Company, St. Louis, Mo.

Vice-Presidents—Mr. George P. Adamson, vice-president and general manager, Baker & Adamson Chemical Company, Easton, Pa.; Dr. Eugene Haanel, director of mines, Ottawa, Ont., Canada; Dr. Leo. H. Baekeland, Yonkers, N. Y.

Secretary—Prof. J. C. Olsen, Polytechnic Institute, Brooklyn, N. Y. Treasurer—Henry S. Renaud, New York. Auditor—Herbert Hollick, Camden, N. J.

Dr. Samuel P. Sadtler and Dr. C. F. McKenna are the past presidents. The directors of the Institute are: H. F. Brown, Wilmington, Del.; Ludwig Reuter, Berkeley, Cal.; Thorn Smith, Detroit, Mich.; J. T. Baker, Phillipsburg, N. J.; Dr. W. M. Grosvenor, New York City; Richard K. Meade, Allentown, Pa.; William M. Booth, Syracuse, N. Y.; Dr. Edward G. Acheson, Niagara Falls, N. Y.; Prof. Edward Hart, Easton.

Notes on Chemistry and Metallurgy in Great Britain.

(From Our London Correspondent.)

The Aëration of Portland Cement.

In a paper read before the Institution of Civil Engineers on Nov. 22 Mr. H. K. G. Bamber reviewed past and present methods of manufacture and advocated the abandonment of the practice of "weathering." He said that many engineers—especially those whose experience extends over a long period—have become so accustomed to the use of some aërating process that, perhaps by force of habit, they still continue to require it of their contractors, forgetting that cement which has already passed the stringent tests of the British standard specification has necessarily shown itself to be entirely innocent of any expansive tendencies. If this specification is adhered to for all requirements, the old aëration methods have become as obsolete as the ancient machinery of the cement industry, which has found its way to the scrap-heap. Contrasting the properties of cement made by the old processes with those of the best modern cements, it should not be difficult to convince engineers of the desirability of abandoning an obsolete method of treatment.

Mr. Bamber might have gone a step further and condemned the most undesirable practice of some cement testers in aërating "hot" samples before making the usual physical tests until the rise of temperature has been reduced very considerably. The writer recalls a case which came under his notice rather more than a year ago, in which an analytical chemist who adhered strictly to the British Engineering Standard Specification condemned a sample of cement as thoroughly unsound; another sample from the same source was sent to a certain professional cement tester, who repeatedly reported that the cement was "still too hot to test," and after aërating it for *more than three weeks*, certified that it was sound. The sequel was that the whole of the work executed with that cement had to be taken out and replaced. If a cement contain such quantity of free calcium oxide as will cause an abnormal rise of temperature on gauging, it cannot be satisfactory; and no amount of weathering or aëration will enable such stuff to compare with properly burnt cement free from excess of lime, because the resulting carbonate and hydrate—sometimes referred to as "hydrated carbonate"—must detract from the cohesive power of the cement.

The Detection of Fire-Damp.

At the Society of Arts, last week, Sir H. H. Cunynghame, in a paper on methods of detecting fire-damp in mines, described a simple addition to the ordinary safety lamp by means of which the "cap" which forms over the flame in the presence of methane or other combustible gas is rendered much more conspicuous. With an ordinary flame the cap is usually so pale that it can only be discerned when the lamp is turned down very low, when it was possible to see the cap in the presence of one-and-a-half or two per cent of methane, and easy to observe it if the air contained three or four per cent.

Professor Cadman and himself devised an improvement which made the cap more visible without unduly diminishing the size of the flame. A small piece of asbestos steeped in sodium carbonate solution and dried was so mounted that it could be brought into the flame of the miner's lamp; and the cap over the resulting bright yellow flame became quite distinct, and rendered the test much more sensitive. He had found that sixty hours' continuous burning was required to exhaust the sodium carbonate in the asbestos; and by actual observation in mines had established the fact that one-half per cent of methane could be seen without any difficulty.

Detection of Inflammable Vapor or Gas.

In a paper recently read before the Institution of Engineers and Shipbuilders in Scotland Mr. J. H. Heck described a method of testing air for petroleum vapors and combustible gases. The apparatus consists of a glass syringe, having a capacity of about four fluid ounces, and a small spirit lamp, giving a flame rather more than one inch in height. The syringe is filled with the air to be tested, and the nozzle is brought to a dull red heat in the spirit flame; then, holding the syringe horizontally, with the orifice close to and slightly below the tip of the flame, the air is expelled by a series of slight jerks of the piston.

An inflammable or explosive mixture ignites and burns at each puff with a bluish color for poor mixtures and a yellowish-blue for those richer. Weaker mixtures of air and vapor give a bright incandescent streak or cap; and air free from combustible gas or vapor produces no incandescent streaks, but only a slight flattening or shrinkage of the top of the flame.

This method does not appear to the writer to be wholly free from risk to the operator. Given a mixture having decided explosive properties, the preliminary heating of the nozzle might, in a bright light, be carried too far and cause ignition with possible disastrous results.

Non-Ferrous Electrometallurgy.

Professor Arnold, in his recent presidential address to the Electrometallurgical Society of Sheffield, said that the Sheffield University had arranged to devote £10,000 to the establishment of a department for research in non-ferrous metals. The University was already in a position to investigate the electrical smelting of brasses, bronzes and German silver, having acquired a Kjellin non-ferrous crucible, with which it was also intended to carry out tests to compare the relative efficiencies of electric melting and coke melting of base metal.

The Relation of Chemistry to Metallurgy.

Mr. A. C. Claudet read a paper on this subject at a meeting of the Society of Chemical Industry, held early in November. He said that the development of the composite science of metallurgy might have been much more rapid during the last thirty years—particularly in respect of some of the non-ferrous metals—if metallurgists had kept closer in touch with the advances in industrial chemistry. Both sciences were allied with mechanical engineering; and frequently the greatest difficulties were those encountered in mechanically working out a process.

As demonstrating the relation between chemical industry and metallurgy he instanced the concentration process employed at mines in the South of Spain on cupriferous pyrites; the treatment of Spanish cupreous iron pyrites, imported by England and other European countries, for extraction of sulphur and copper; and the treatment of lead-zinc-silver sulphide ores from Broken Hill. The manufacture of copper sulphate might at first sight seem to be purely a chemical process; but it involved various important metallurgical operations. The Pearce process for auriferous copper was another example.

The arsenical cobalt and nickel silver ores are now worked in Ontario, affording a promising field for chemical and metallurgical research. Their silver content was very large—often thousands of ounces to the ton; but their satisfactory treatment presented great difficulties, for the arsenic sometimes amounted to thirty per cent, and the cobalt and nickel to ten or fifteen per cent each, and a method of recovering all these as well as the silver was wanted.

Market Prices in November.

Copper rose to £57.16.3 on the 2d, remained steady until mid-month, and gradually declined to £57.5.0 on the 30th.

Tin opened at £167.10.0, dropped to £166 on the 9th; then recovered the same figure by the 23d, and rose to £169 at the month's end.

Lead, except for a slight decline on the 9th, kept at about £13.12.6 until the last week, when it declined to £13.11.3.

Haematite fell from 65/3 on the 2d to 64/- on the 16th, and improved to 64/6 on the 30th.

Scotch Pig opened at 55/10½, then fluctuated with a minimum of 55/4½, and closed at 55/9.

Cleveland Warrants from 49/10 fell to 49/5½ on the 9th, with but slight improvement until the end of the month, when the quotation was 49/9.

Alum, lamp, loose, per ton	£5.12. 6.
Antimony, black sulphide, powder, per ton	22. 0. 0.
Borax, British refined, crystal, per ton	16. 0. 0.
Sulphate of ammonia, f.o.b. Liverpool, per ton	12.16. 3.
Sal ammoniac, lamp, firsts, delivered U. K., per ton	42. 0. 0.
Copper sulphate, per ton	19.10. 0.
Caustic soda, ash 48 per cent ordinary per ton	5.10. 0.
Litharge, flake, per ton	15.15. 0
Bleaching powder, 35 per cent, per ton	4. 5. 0.
Sulphur, recovered, per ton	5. 5. 0.
Camphor, 1 oz. tablets	1. 7.
Shellac, standard T. N. Orange spots, per cwt	4. 3. 0.
Hydrochloric acid, per cwt	5. 0.
Copper ore, 10 to 25 per cent, unit	9/4½ to 10.4½.
Tin ore, 70 per cent, per ton	£104 to 106. 0. 0.
Carbolic acid, liquid, 97/99 per cent, per gal	1. 1. 1.
Creosote, ordinary good liquid, per gal	2.
Naphtha, solvent, 90 per cent at 160 deg. C., per gal.	10.
Mica, small slab, original cases, per lb	6d to 1. 6.

India rubber: Paraffine, at the middle of the month prices had fully recovered from the depression at the end of October; and during the last week there has been an advance of 2d. to 3d. per lb.; prices being now steady at 6/3.

Platinum has risen to £8 per oz.

Synopsis of Chemical and Metallurgical Literature

Chemical Engineering.

Evolution of Portland Cement Processes.—In his presidential address to the American Institute of Chemical Engineers Dr. CHAS. F. MCKENNA gave a rapid review of the history of Portland cement from the day of Aspdin's invention, portraying particularly the different stages in the process from the crude method of Aspdin to the modern process of burning in rotary kilns and the fine pulverizing. The economy of the present day is such that for 1 cent 12 lb. of raw material can be taken from the earth, dried, ground, calcined and vitrified and the product ground again. Considerable attention was paid to the old forms of wet processes, and the statement was made that present-day European practice in grinding to semi-wet slurry hard materials, not marls, is the most important advance in modern times and should be taken up more vigorously in America. The basis for this is the claim that the dryer expense is saved, finer incorporation is obtained, better clinkering and easier grinding of the clinker. The critical feature of the modern semi-wet process is to secure the maximum of fineness of slurry with the minimum water. It was said that in good practice some materials can make with less than 35 per cent of water a slurry which can be pumped, and as it dries in the upper end of the long kiln the extra coal consumption is not marked. Modern rotary kiln practice, particularly the long kiln, was touched upon, and the use of preheated air from clinker kilns and diminished indraft of cold air.

Hydrated Lime.—The manufacture of hydrated lime was discussed in a paper of Mr. RICHARD K. MEADE before the American Institute of Chemical Engineers. Hydrated lime is mechanically slaked lime. Just the correct portion of water is added so that the resulting product is a dry fluffy powder. Most of the hydrated limes on the market up to a few years ago were very crude products, usually containing considerable carbonate. Now, however, processes have been perfected which produce a lime consisting almost entirely of the hydrate. The process for the manufacture of hydrated lime usually consists of three stages, (1) crushing the lime, (2) mixing the lime and water, (3) separating the hydrate from the unhydrated portion. In some processes instead of separating the hydrate from the unhydrated portion the two are ground together in some form of pulverizing mill.

The hydrating plant itself is usually located adjacent to the limekilns. The lime is drawn from the kilns and is immediately crushed in some form of rotary or pot crusher. From this it is conveyed mechanically to the hydrator, in which the slaking is done. There are several forms of this latter in use, but all come under two heads, continuous hydrators and batch hydrators. The continuous hydrator most used is the Kritzer hydrator, which consists of a number of cylinders mounted one above the other on a steel framework. Through the middle of each cylinder runs a shaft to which are fastened paddles. The lime is fed in at the upper cylinder and mixed with the water. The paddles stir it up and also force it to travel through the cylinder. It then drops into the next cylinder and here it is mixed up and passed on by the paddles in this and so on until it has passed all of the cylinders. The proportions of lime and water are determined by means of an automatic feed for the lime and a valve for the water.

The Clyde hydrator is a batch hydrator in which the lime and water are weighed in separate charges consisting of about 1500 lb. of lime and the proper amount of water for this much lime. The pan rests upon a ball bearing and is revolved by means of a gear and pinion on the under side of the pan, which is surmounted by a hood and a stack. Plows running across the surface of the pan are fastened to a stationary framework above the latter. The lime and water are fed into the hydrator through an opening in the side, and the plows stir it up and mix the two. When hydration is completed the lime is discharged through an opening in the center of the hydrator, which during the mixing is closed.

After passing through either hydrator that portion of the lime which has been slaked is in the form of a very fine fluffy powder all of which will pass through a very fine sieve. Mixed in with this, however, are always some unslaked lime, unburned limestone and silicious cores. For the better grades of hydrate it is usual to separate out this foreign material by means of screens, although where the amount of these constituents is small they are sometimes ground in with the hydrate. Fuller-Lehigh and Raymond mills are principally used for the grinding, while the separators most used are the Newaygo and the Columbian.

The packing of the hydrate is best done in cloth (100 lb.) or paper (40 lb.) bags automatically by means of the Bates system, which consists of a valve bag and machine for filling the same. The bags are provided with a valve and are tied before being filled. This valve is made by ripping, folding and sewing one corner of the bag. The cement is inserted through the valve by means of a tube on the bagging machine, which automatically fills and weighs the bag. The machinery for hydrating magnesian lime having a capacity of two tons or three tons per hour costs about $5,000 and for high calcium limes $12,000. The cost of hydrating lime is about 25 cents to 30 cents per ton. One ton of lime, however, will often make more than one ton of hydrate.

The advantages of hydrated lime are: It is a fixed product and does not deteriorate with age; it can be handled without risk of fire; it may be shipped by water and packed in cloth or paper bags; it can be used for all purposes for which lime is used and is more convenient than the latter. In addition to this it can be used for mixing with cement, for which purpose quicklime cannot be used.

Bleaching of Oil with Fuller's Earth.—This subject was discussed at considerable length in a paper before the American Institute of Chemical Engineers on Dec. 8 by Mr. DAVID WESSON. The literature of Fuller's earth and its application is very meager. It appears to be first mentioned about 1889. From the best information obtainable Fuller's earth was first used for bleaching fats and oils in Chicago on a large scale

about 1879, or 1880, although it is said to have been used for a long time previous in northern Africa for clarification of olive oil. The first application was very cumbrous, the earth being stirred into the hot oil and allowed to settle, after which the oil was decanted and the residue boiled with water and pressed. The filter press was first used between 1880 and 1882. The efficiency of a given sample of Fuller's earth cannot be readily determined by chemical analysis. The best method is by making a bleach test on a small scale. In comparing the bleaching properties of two earths they should always be passed through sieves of the same size in order to obtain an accurate comparison. Fuller's earth doubtless owes some of its bleaching properties to the presence of a certain amount of water of constitution. This, if driven off by either drying of the earth or by adding it to oil of too high a temperature, will prevent bleaching. The reason why Fuller's earth seems to bleach is not positively known. Fuller's earths differ greatly in their absorptive properties and show great differences in their liability to spontaneous combustion. The arranging of filter plants is a matter of considerable importance. The best place for filters is on the top of a refinery under suitable shelter. By having them at an elevation above the bleach tanks oil can be allowed to run back to the same, or can be fed to any part of the building, or run easily to storage by gravity; besides, the fire risk is much less, and when the filters are steamed out the nuisance of the steaming is entirely done away with. The mixing kettles for oil and Fuller's earth are practically cylinders provided with mechanical agitation and suitable heating coils. Filter pumps should be slow-moving, capable of working up to a pressure of 150 lb. to 200 lb. The operation of the filter plant is quite simple. The earth is mixed with the oil at the proper temperature and then pumped through the filter press, the first portions of the oil being allowed to run back to the mixing kettle, or bleacher, as it is often called, until it comes up to the required standard, after which it runs directly to storage.

Fitzgibbon Boiler.—In a paper read before the American Institute of Chemical Engineers Mr. Jerome Alexander gave a brief description of the construction and operating principles of the Fitzgibbon boiler, a type combining vertical with horizontal tubular features. The boiler is internally fired, and is entirely without external or internal brickwork, thus avoiding the annoyance and expense of repairs at inopportune times. A heavy asbestos or magnesia covering reduces radiation below that found in ordinary bricked-in boilers. The boiler is a quick steamer, economical in coal consumption, and can be easily moved about.

Sewage Disposal.

The problem of sewage disposal was the subject of a symposium of papers presented at a joint meeting of the New York Section of the American Chemical Society and of the American Institute of Chemical Engineers at the Chemists' Club in New York City on Dec. 9.

Basic Principles.—Mr. Geo. C. Whipple pointed out in his paper that the basic principles underlying all methods of sewage disposal are to get rid of sewage without danger to the public health, with the least possible nuisance to the smallest number of people, with the least damage to property, and at the smallest cost. The process of purification is usually carried on in several stages: First, a partial removal of the suspended matter by screening or sedimentation, or both; second, a process of oxidation using sprinkling filters, contact beds or sand filtration, and, third, a final removal of bacteria by sand filtration or disinfection. The term "sewage purification" is popularly applied to any or all of these processes. This has contributed not a little to confusion of ideas. Laymen innocently suppose that when sewage is "purified" it becomes pure, whereas the sanitary engineer may mean only that it is purer than it was before. How much purer depends upon the method used. It would be of decided benefit to the question of sanitation if more definite terms were used, such as sewage clarification, oxidation, deodorization, disinfection, filtration, etc.

The chief danger from sewage is the contamination of public water supplies. For the protection of public water supplies it is not safe to depend upon sewage purification works alone unless more than ordinary precautions are taken, for the reason that most of the processes cannot be depended upon to remove bacteria from the sewage to a sufficient extent, while with the combined system in use complete purification is impossible. Sewage purification is of advantage, as it offers a second line of defense, an increased factor of safety. It also serves to prevent nuisances to sight and smell, which are of real damage to property and affect personal comfort. Sewage works themselves, however, may be the cause of similar nuisance.

What appears to be needed in this country at the present time is some method of co-operation by which needed sanitary reforms can be brought about at least expense. It is unbusinesslike to compel the purification of the sewage of a large upstream city in order to protect the water supply of a small city lower down, provided pure water can be furnished the latter in some better and cheaper way. Legislation that clothes the State authorities with power to prevent the pollution of sewage, but does not give them power to compel the purification of water or to control pollution by trade wastes, is unfortunate. It naturally leads to litigation rather than co-operation and may retard rather than hasten necessary sanitary reforms. If our State authorities cannot be trusted in this matter it may be that a proper solution of the difficulty will be found in the establishment of district boards similar to those in England and Germany, such boards having jurisdiction over the limits of particular watersheds. In some respects these natural hydrographic boundaries have advantages over artificial State boundaries. In the near future also our national government will take a hand in the matter. In whatever form the authority may be constituted the idea of co-operation should prevail, and ironclad rules against stream pollution should give way to a rational distribution of the burden of purification of both water and sewage, and an equitable adjustment of cost made between the parties interested, thus decreasing the total expense of sanitary measures required and utilizing natural resources for the purification of sewage in water as far as this is safe.

If the system of water carriage of sewage continues in use the time will some day come when the sewage of all of our cities will be purified, partially or completely, and all surface water supplies filtered. It is proper to anticipate this consummation as far as our means permit, but meantime it is good business and sound common sense to spend our money first where it will go furthest and do the most good, building water filters and sewage purification works, sometimes one, sometimes both, as they may be needed. Adequate remedies against stream pollution from the standpoint of nuisance have been usually obtained by an appeal to the principles of common law. Cases involving bacterial pollution by sewage have been thus far too few to establish definite precedents. It will be interesting to see whether, in view of our increasing population, and especially the increasing growth of our cities, the courts will ultimately decide that the use of unfiltered river water as a source of water supply by riparian owners is a reasonable use of the water. The speaker was optimistic in regard to the improvements that are being made in methods of sewage disposal. He believes that the near future is to see extensive developments in the art. Important as are the recent improvements in the technical details, still more important and necessary is a rational assignment of the means employed to the work at hand. The use that is to be made of the water into which sewage is discharged and the conditions tending to natural purification are to be studied with greater minuteness than ever before; the method of dilution is to be used more

rationally, and the designs of the sewage disposal plants made to fit more closely the particular conditions. The general result will be the construction of more and simpler plants for the treatment of sewage and, where occasion makes necessary, the installation of yet more efficient devices for bacterial purification than those now employed.

Sewage Disposal in Europe.—Dr. RUDOLPH HERING gave in his paper a review of European practice. Disposal of sewage by dilution was first practised. Of the 16 largest cities in the world having a population of over 1,000,000 inhabitants every one without exception now employs the dilution method where flowing water is available and a sufficient quantity of oxygen is dissolved in it to represent a flow of fresh water equivalent to at least 3 cu. ft. per second for every 1000 persons discharging sewage. Where a water supply is taken from a river it must be purified even after any sewage entering above has been treated. Where a river is not used as a water supply sewage may contaminate the food fish, oysters, etc. The commercial value of such food is insignificant compared with the cost of removing all the conditions that would impair their hygienic qualities. It is therefore much cheaper to take them at a greater distance from the large cities where no dangers exist.

The nuisance to sight produced by sewage is almost everywhere in Europe prevented by keeping all plainly visible floating matter out of a river by screening. Screens with a mesh as small as 1 mm are used. The nuisance to smell is prevented by constructing sewers with such smooth interior surface and good grade as will deliver the sewage practically fresh to the outfall and insure an immediate submergence of the sewage as far below the surface of the water as possible. This method is entirely successful in many Continental cities. The amount of sewage discharged into the Thames is so regulated that the dissolved oxygen in the water does not generally fall below 30 per cent of saturation. No nuisance results under these conditions. European experience justifies the conclusion that purification of sewage by the oxygen in flowing water is entirely satisfactory.

Sand purification of sewage requires the delivery of the sewage in fresh condition and a preliminary separation of the coarser suspended particles, after which the finely divided matter may be oxidized in intermittent sand filters. The more recent developments involve improvements on the septic tank method of purification. Dr. Imhoff has developed a tank having two compartments. In the upper the suspended matter settles out and is drawn off through a slot. In the lower compartment the sewage is decomposed with the evolution of methane, carbon monoxide, and dioxide. No sulphuretted hydrogen is produced. The decomposition is complete in from three to six months. The sludge which is removed dries quickly to a friable material similar to garden mold. The annoying question of the disposal of sewage sludge may therefore be considered satisfactorily settled.

Sewage Disposal in New York.—This was discussed in a paper by Dr. GEO. A. SOPER. For the most part the sewage of New York is conveyed in crude condition by the shortest route to tide water and discharged at or near the surface of the water. There are over 500 sewer outlets discharging 600,000,000 gal. of sewage per day. In all parts of the harbor above the Narrows the water is seriously polluted and rendered objectionable both to sight and smell. Along 175 miles of shore the water is so polluted as to be dangerous to bathe in. These conditions will rapidly become worse as the population increases. A standard of purity for the water of New York harbor has been proposed by Colonel Black and Professor Phelps as a result of studies, estimate and apportionment. They consider that the dissolved oxygen in the water should not fall below 70 per cent of what would be present if the water did not receive any sewage. This limit has already been exceeded. A definite plan and program of main drainage, including outfall works and purification plants designed to improve the present unsanitary conditions, have been proposed by the Metropolitan Sewage Commission. The duty of carrying out this plan so far as New York is concerned has been intrusted to this commission. This work is now progressing satisfactorily.

Unsolved Problems.—Prof. C. E. A. WINSLOW discussed in his paper unsolved problems of sewage disposal. The three main ends of sewage treatment—the elimination of suspended solids, the oxidation of organic matter, and the destruction of pathogenic bacteria—can all be attained by practically efficient methods, but there are minor difficulties to be overcome in each case. The pathogenic bacteria can be destroyed when necessary by disinfection with bleaching powder, but the dosing devices available for use in small plants are very defective. The oxidation of organic matter can be economically effected by the use of sand, contact or trickling beds. Distribution systems for trickling filters require further study. The intermittent sand filter has possibilities of intensive action not attained in plants operated at present largely at haphazard. It is the accumulation of suspended solids on the surface which keeps the capacity of sand beds down to 100,000 gal. per acre per day and experiments carried out at the Sewage Experiment Station of the Massachusetts Institute of Technology show that the rate can be raised to 200,000 gal. or 300,000 gal. if suspended solids are removed by careful preliminary treatment. Sludge disposal is still the most serious of the unsolved problems of sewage treatment. It has been minimized at the Technology Experiment Station by the use of a deep tank on the septic principle, but so operated that the sewage rises and washes the sludge continually, and thus keeps it from becoming overseptic. By this means the liquefying process is so accelerated that over 70 per cent of the deposited total suspended solids have been liquefied. The utilization of sewage sludge, though not yet economically realized, furnishes promising possibilities to the industrial chemist in the future.

MR. NICHOLAS S. HILL, JR., presented a paper on sanitary conditions in their relation to water supplies in the vicinity of New York City, touching upon the principal legal questions involved in the problem.

Recent Metallurgical and Chemical Patents

Gold and Silver.

Clancy Cyanide Process.—The details of the improvements which have been made in the cyanide process by JOHN COLLINS CLANCY were given in our issue for November, 1910, page 623. They are again mentioned here in order that those desiring to secure copies of the United States patents may do so. (976,043-4-5, Nov. 15, 1910.)

Zinc.

Production of Zinc Powder.—In the process of first making zinc oxide and then reducing the oxide in a separate process, OLIVER B. DAWSON, of El Paso, Tex., proposes to introduce into the zinc vapor, at a point just above or close to the bed of the mixture under treatment, sprays of water gas. The object of this is to supply carbon monoxide to combine with oxygen present, either free or combined with zinc, forming carbon dioxide, and leaving metallic zinc in the form of fume which is subsequently condensed. (976,557, Nov. 22, 1910.)

Rare Metals.

Pyrophoric Mass.—An improved invention relating to those substances which give off sparks of great intensity when rubbed with a metallic substance has been patented by DR. CARL AUER VON WELSBACH, of Vienna. These substances are of value in igniting liquids, solid and gaseous bodies, as well as for the purposes of illumination, signaling and photography. The inventor has discovered that cerium, or metals of the cerium group, may be treated to form a pyrophoric body which is hard and durable, resisting the decomposing action of air.

Since the pyrophoric property of the metal seems to lie in its suboxides or nitrides, the procedure is as follows: The metal is first reduced to fine powder, then compressed into suitable molds by a pressure of from 1000 kilos to 25,000 kilos per square centimeter. This mass is slightly pyrophoric, but the property is further increased by heating with an equal volume of air in a metal casing having thin walls. The temperature is raised gradually to incipient red heat, when a lively reaction ensues and the mass frits together. The fritted mass is stable and is so highly pyrophoric that when a knife-blade is drawn across it an actual sheet of flame is produced. For varying the hardness of the alloy certain other metals such as zinc, tin, lead, calcium, magnesium and aluminium may be added. (976,760, Nov. 22, 1910.)

Melting Furnaces.

Crucible Furnace Cover.—Furnaces adapted to the melting of metals in crucibles are not always provided with covers which prevent the flame of the furnace coming in contact with the metal, thereby oxidizing it and causing loss. It is the object of a recent invention by Julian L. R. Brown, of Bridgeport, Conn., to provide a cover for a furnace, which shall at the same time act as a cover for the crucible and prevent the deterioration of the metal by the action of the flame. The device will be understood by reference to Fig. 1, which shows a plan of the cover and a section of the furnace. The cover may be made of any suitable material, such as fire-clay, graphite or brick. The essential features are the central portion adapted to cover the crucible, and escape holes for the flame, in alignment with the flame gap surrounding the crucible. Various modifications of the idea are also patented. (975,506, Nov. 15, 1910.)

Charging Melting Furnaces.—In practice there is considerable wear on melting furnaces, and especially on electric furnaces, by throwing in the rough scrap and metal, which digs into the sides of the furnaces and weakens them. To obviate this difficulty Michael R. Conley, of New York has patented a cartridge of sheet metal adapted to fit the melting pot of the furnace and contain the charge to be melted. The entire surface of the cartridge comes in contact with the

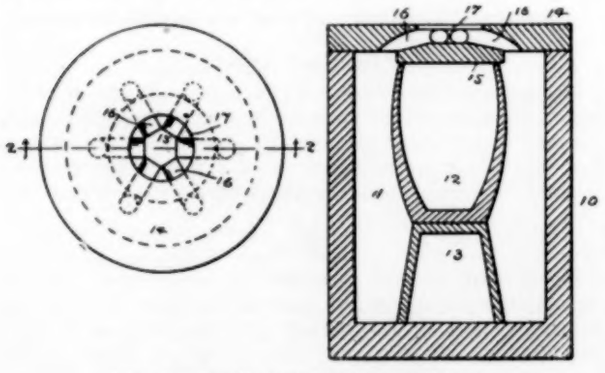


FIG. 1.—CRUCIBLE FURNACE WITH COVER.

hot furnace wall and the charge begins to melt evenly and quickly. In practice the cartridge may be made of any desired composition, and thus not disturb the nature of the melted product. (975,751, Nov. 15, 1910.)

Reversible Heating Furnaces.

Furnace Head for Gas Furnace.—Relating to furnaces in which gas and air are simultaneously introduced and in which the stream of gas and air is periodically reversed, Bruno Versen, of Dortmund, Germany, has patented an improved furnace head, illustrated in Fig. 2. An object of the invention is to improve the caloric efficiency of the furnace by providing apertures in the furnace head for the escape of the "heat," i. e., the combustion mixture of air and gas, said apertures being at about the same height as the apertures for the introduction of the gas. By this means the air which served for raising the temperature of the heating flame is maintained within the said flame in the entire interior space of the furnace.

In the improved head air is conveyed by vertical lateral flues *L* which open into the top of the furnace by a common port *L'*. Gas is allowed to enter by a flue between the air flue and by a port below the common air port. Below the latter branch flues *Z* extend from the lateral flues *L* and open into the furnace as the gas flue *G*. When the head is acting as an exhaust, the heat on the hearth of the furnace will be conveyed through *Z*. When the head serves as an intake the blast of

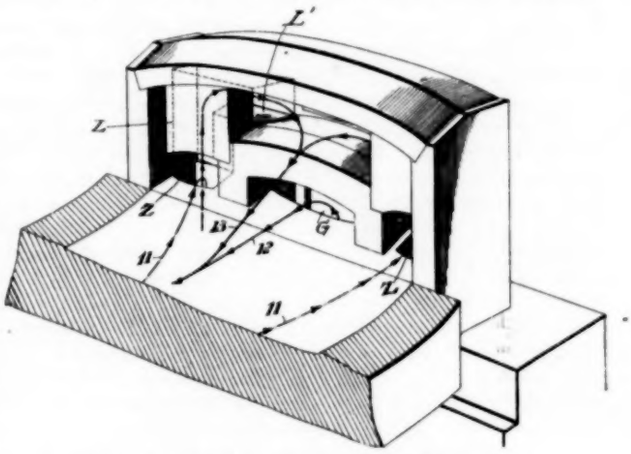


FIG. 2.—FURNACE HEAD FOR GAS FURNACE.

air will come in through *L*, and owing to its momentum will pass *Z* and enter through *L'*. For the purpose of protecting the furnace head against undue deterioration the inventor places cooling pipes in various portions of the head, not shown in this illustration. The system of piping also serves as a reinforcement, resisting vibrations and jars. On account of the advantageous distribution of the air the weight of the head may be decreased, so that even in very large furnaces the head may be easily removed. (975,738, Nov. 15, 1910.)

Roasting Furnaces.

Roasting Sulphides or Sulphates.—In a furnace of the superposed hearth type, Utley Wedge, of Ardmore, Pa., has introduced a modification designed to facilitate the elimination of sulphur from ore, particularly in the latter part of the roast. In the drawing, Fig. 3, is shown a furnace of the type mentioned, with the addition under consideration. The furnace is divided into two sections, upper and lower. In the upper section the roasting operation proceeds as usual, but on coming to the lower section the ore is fed through a seal formed by a shelf below the passage from upper to lower sections. The ore is scraped from the shelf by an arm attached to the upper rabble of the lower section. Between the arch of the upper chamber of the lower section and the hearth of the lower chamber of the upper section is provided means for introducing a carbonaceous reducing agent such as coke breeze or pulverized coal. This will reduce any sulphate compounds which may have been formed and will permit the elimination of the remaining sulphur by the application of a temperature approximating 1200° F., as compared with 1600° F. or 1800° F. commonly required in the lower portion of the furnace as usually operated. Arrangement is made to permit of either mixing the gases from both sections, or conveying them from the furnace separately. By preference the lower section of the furnace is of the muffle type, the heating chambers being supplied with the products of combustion from a heating furnace shown at the side. (976,525, Nov. 22, 1910.)

Roasting Furnace.—An improvement in his well-known roasting furnace has been patented by John B. F. Herreshoff, of New York. The object is to improve the life and efficiency

of the apparatus by proper cooling arrangement. The hollow shaft of the furnace is closed at its upper end and has a surrounding sleeve and rabbles attached to the shaft. Connection is made between the rabbles and the shaft and sleeve, whereby air may travel upward through the shaft, thence outward into each rabble arm and back into the sleeve, which is open at the top. By this circulation of air, which is driven by a blower, the hot parts of the furnace are cooled. (976,175, Nov. 22, 1910.)

Annular Roasting Kiln.—A furnace of this type with a number of novel improvements has been patented by JOHN

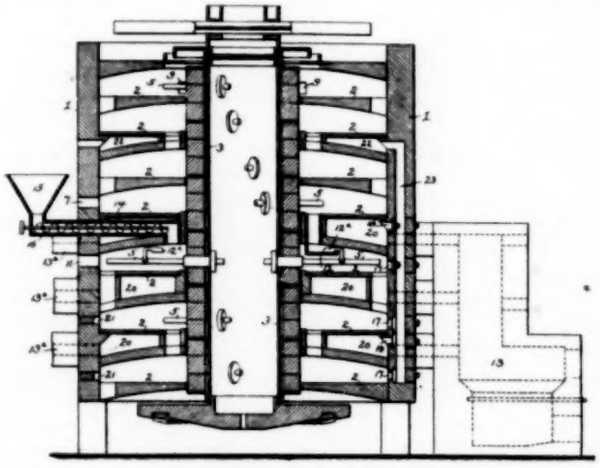


FIG. 3.—ROASTING FURNACE.

ZELLWEGER, of St. Louis, Mo. The principal features are means for recuperating heat from the fire gases; the use of warm air for oxidizing the charge; separation of sulphur gases from the products of combustion; and variation of temperature in the several zones of the hearth, whereby the ore may be locally heated to different degrees of temperature. (976,769, Nov. 22, 1910.)

On account of limitations of space a very large portion of Synopsis and Patent Abstracts had to be reserved for our February issue.

Vacuum Evaporator.

Evaporators are employed in many industries, often in the recovery of valuable constituents of solutions ordinarily considered worthless; frequently, too, unutilized exhaust steam occurs. In such cases a multiple-effect evaporator operated by this exhaust can be utilized to the greatest advantage. As an instance, it is only within the past few years that the tank-water which occurs in packing houses, etc., has been evaporated and its valuable solids recovered.

This tank-water averages from 5 per cent to 7 per cent solids; the smallest packing and rendering plants produce several hundred gallons daily and the value of the solids contained averages $35 per ton, whereas the cost of producing them, by means of the multiple-effect evaporator, will not exceed $10 per ton. An evaporator should, therefore, pay for itself in from six to eight months, depending upon the quantity and quality of the water.

The following notes refer to the construction and uses of the Redfield "standard evaporator," built by Messrs. Robert S. Redfield & Company, 90 West Street, New York City, for the evaporation of tank-water, glue, gelatine, garbage liquors, spent lyes, caustic potash, beef extract, cane and beet sugar, glucose, tanning extract, brine copperas, and many other special liquors occurring in chemical works and other establishments.

The Redfield standard evaporators are of the vertical tube type with circular steel shells or square cast-iron shells, both types being fitted with removable heat chambers, built with copper, steel or charcoal-iron tubes, as necessity requires. The entire heating chamber is removable en bloc from the body of the evaporator.

The shells are fitted with manhole and side glass fixtures, affording at all times an easy access to and a clear view of the interior of the shells.

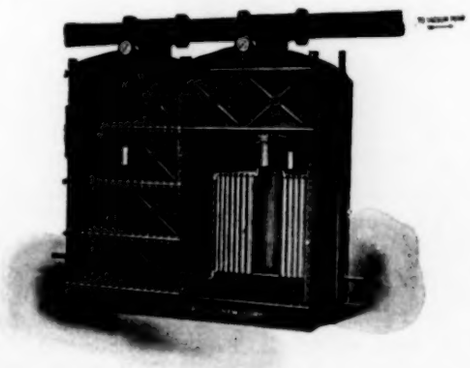
The fundamental distinction between the Redfield standard and other types of evaporator lies in the fact that in the Redfield apparatus the steam is outside instead of inside the tubes. The liquor therefore is afforded a free circulation upward through the tubes and returns to their base by way of the open space between the shell and the steam drum proper.

When the steam is caused to enter the steam chamber the heated liquors at once rise in the vertical tubes. There is nothing to impede this movement and with increasing temperature and higher vacuum the movement of the liquor becomes so rapid that jets will be thrown up often to a height of 2 ft. or 3 ft. It is obvious that this hot liquor must in some manner return to the base of the evaporator, which is cooler than the tube passages from which it emerged. This passage is provided by the space between the steam drum and shell, with the result that there is a constant and exceedingly rapid and automatic circulation of the water rising through the tubes, thence around the heating chamber to the base of the tubes.

By virtue of this vertical-tube construction the tubes automatically clean themselves due to the extremely rapid passage of the liquor throughout their length while the high speed at which the liquor circulates entirely prevents incrustation, resulting in freedom from tube repairs and saving in steam.

In the latest type of the Redfield standard evaporator the steam in each effect enters the center of the steam chamber. This construction at once insures a perfectly even steam distribution, which in part accounts for the remarkable reserve capacity. The condensation being automatically and immediately removed the full efficiency of the steam is constant. For this reason this type of evaporator is particularly well adapted for such liquors (like glue and gelatine) where quick evaporation is required, as too long contact with hot surfaces would deteriorate the quality of the product.

The tubes in the steam chamber are expanded into the tube



EVAPORATOR.

sheets by the same method that boiler tubes are expanded into the tube sheets of high-class boilers.

If steel tubes are employed they have a practically unlimited life. Copper tubes, owing to the great expense, are never used heavier than 16-gage, and these light tubes cannot be kept tight for any great period of time nor can they stand up as would a heavier tube. Practically the only other maintenance expense is on the pumps.

The fact that the steam chambers in the Redfield standard evaporators are entirely separate from the main body permits the removal of the entire steam chambers and the replacement of steam chambers with from 50 per cent to 100 per cent increased capacity.

These evaporators are virtually automatic in operation. The feed valves may be so regulated that a constant supply of fresh tank-water is drawn in to counterbalance the rapid evaporation. Each plant is provided with a stick testing funnel so constructed that a quantity of the concentrated liquor may be withdrawn from the last effect quickly and easily to be tested for density. Opening a valve at the base of this funnel allows the vacuum to pull the tested liquor back into the evaporator. This method prevents all loss of concentrated liquor and is extremely simple.

An entire evaporating plant, either single or multiple effect,

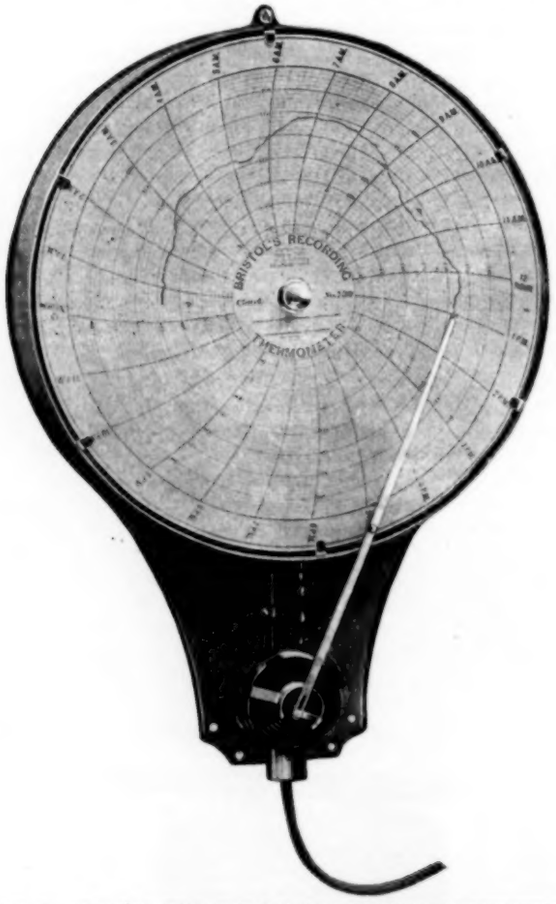


FIG. 1.—INTERIOR CONSTRUCTION OF RECORDING THERMOMETER.

may be easily handled by ordinary labor. In plants up to 2000 gal. only a part of one man's time is required.

To save floor space in multiple-effect evaporators all space may be eliminated between different effects. The length of the large vapor piping as well as all other piping connected with a multiple-effect plant of this type is reduced to a minimum.

A double-effect Redfield standard evaporator is shown in the adjoining diagram. Messrs. Robert S. Redfield & Company build their standard evaporators in any capacity from 100 gal. to 5000 gal. per hour.

High-Pressure Tanks.—For containing gases or liquids under pressure cylinders that are merely riveted and calked have often proved unsatisfactory; such a cylinder may even be perfectly tight under a high-pressure water or liquid test, yet may be found to leak badly when only a slight air or gas pressure is applied. For this reason brazed tanks or cylinders are now generally preferred for high-pressure tanks. With a brazed tank it is well known that the brazed joints or seams are the strongest parts, as they form a perfect union when properly brazed. Concerning such brazed high-pressure tanks interesting information may be found in the catalog of the manufacturers, the Wm. B. Scaife & Sons Co., of Pittsburgh.

Compensated Gas-Filled Recording Thermometers

During the last 15 years recording thermometers of the Bristol Company, of Waterbury, Conn., have been constructed in various different forms, depending for their operation on the expansion of a liquid, the expansion of the vapor of a liquid, or the expansion of a gas. These thermometers have been used very extensively for ranges of temperature up to 800° Fahr., but the models equipped with flexible connecting tubes between the sensitive bulb and recording instrument and depending for their operation on the expansion of a vapor or a gas have not until recently been adapted for recording the lower ranges of temperature.

A new compensated gas-filled Bristol recording thermometer has recently been developed for recording the lower ranges of temperatures, such as atmospheric temperatures, temperatures of water, temperatures of brine in refrigeration systems, etc., and found satisfactory in numerous tests. These thermometers are equipped with a patented compensating device which automatically corrects for changes of temperature at the recording instrument.

Bristol's gas-filled class III recording thermometers are equipped with sensitive bulb and flexible capillary connecting tube and a pressure tube, as shown in Fig. 1, the sensitive bulb and flexible connecting tube and spiral pressure tube all being filled with an inert gas under pressure. Changes of temperature at the sensitive bulb cause corresponding changes in the pressure of the confined gas and these changes in pressure are measured and recorded by the recording instrument. The sensitive bulb is usually about 10 in. long and ¾ in. in diameter and the volume of gas contained in this sensitive bulb is very large in proportion to the volume of gas contained in the fine capillary connecting tube between the sensitive bulb and the recording instrument, this making the error due to changes of temperature along the connecting tube negligible.

Scientists have long recognized the fact that an air or gas

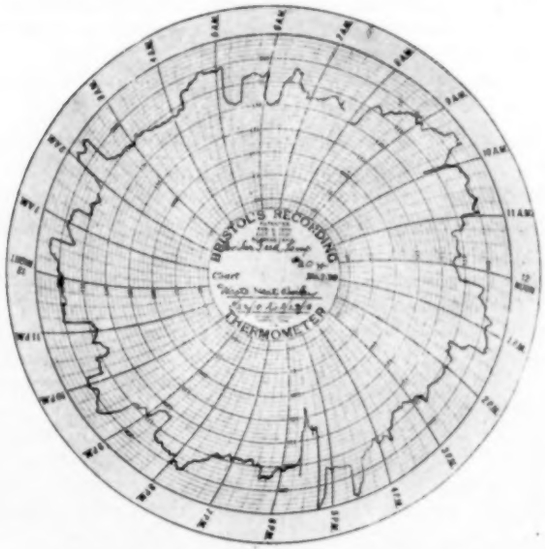


FIG. 2.—RECORD OF FEED WATER TEMPERATURE.

thermometer is the ideal type of apparatus for measuring low temperatures, provided that the air thermometer can be manipulated in the laboratory, where it is possible to make various corrections. The very important new feature of the Bristol compensated gas-filled recording thermometer is the patented compensating attachment for the spiral pressure tube, shown in Fig. 1, since a thermometer equipped with this compensator gives the same readings or record when the temperature at the recording instrument changes as it would if the temperature at the recording instrument remained constant.

The need for such a compensator can be illustrated by the

application of a recording thermometer for recording temperature of brine in a refrigeration system. The temperature of the atmosphere at the point where the recording instrument was installed might change, although the temperature of the brine at the point where the sensitive bulb was installed remained constant and a recording thermometer for brine temperature should, of course, be so constructed that it would be affected only by changes of temperature at the sensitive bulb.

Fig. 2 shows a record of feed water temperature as recorded by one of these instruments, the chart having range of 40°

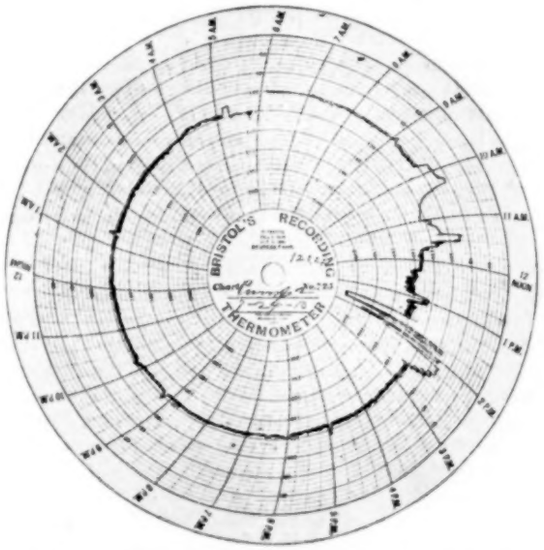


FIG. 3.—RECORD OF CONDENSER WATER TEMPERATURES.

Fahr. to 225° Fahr. It will be noted that the scale of this chart has uniform graduations over the entire range and this makes it possible to use the Bristol-Durand radii averaging instrument for computing the average of the feed water temperature for the 24 hours.

Fig. 3 shows a record of condenser water temperatures, as recorded by one of these thermometers on a chart having range of 40° Fahr. to 220° Fahr. It will be noted that this chart has a reversed scale of uniform graduations. For condenser water temperatures the reversed scale is a desirable feature, as the average temperatures are usually lower than 125° Fahr., whereas the temperatures are likely to reach boiling point occasionally, at times of loss of vacuum in the condenser, as shown by the record in Fig. 3, between 1:30 p. m. and 1:45 p. m.

American Museum of Safety.—The formal exercises in connection with the permanent exhibition of safety devices of the American Museum of Safety were held on Nov. 21 at the Auditorium of the Engineering Societies Building, 29 West Thirty-ninth Street, New York. The object of the Museum is the conservation of human life by means of a permanent exhibit of the best and most practical safety devices for making safe the dangerous parts of machines and processes. Mr. Philip P. Dodge presided. Among the speakers were the director of the Museum, Dr. W. H. Tolman, who discussed Germany's system of accident prevention; Prof. F. R. Hutton, Mr. T. C. Martin, Mr. E. S. Lott and Dr. N. E. Ditman. Mr. S. C. Dunham, president of the Travelers' Insurance Company, presented the gold medal offered by his company to the individual industrialist or corporation that has done the most for the protection of the lives and limbs of their workmen by means of safety devices for machines or processes to the United States Steel Corporation. Mr. W. B. Dickson, first vice-president, accepted the medal on behalf of the Steel Corporation.

Personal

Dr. **Leo H. Baekeland,** of Yonkers, N. Y., has been awarded by the City of Philadelphia, on the recommendation of the Franklin Institute, the John Scott Legacy Premium and Medal for his invention of bakelite, in accordance with the report of the Institution's committee on science and arts, adopted May 4, 1910.

Mr. **Walter Brinton,** superintendent of the manganese steel department of the Taylor Iron & Steel Company's plant at High Bridge, N. J., since 1895, has resigned and has accepted a position as consulting engineer for the Edgar Allen American Manganese Steel Company, which is manufacturing manganese steel at Chicago Heights, Ill., and at New Castle, Del. Mr. Brinton's headquarters will be at the New Castle plant.

Dr. **Allerton S. Cushman** has resigned his position with the United States Department of Agriculture to found the Institute of Industrial Research in the City of Washington. The object of the Institute will be to undertake investigations of industrial problems in all lines, but more particularly of investigations concerning the protection and conservation of industrial metals; agricultural, land and fertilizer problems; hydraulic cements; paint technology; chemical industry, chemical engineering and electrochemistry. The Institute will be located in the heart of the City of Washington, near the White House, and fronting upon the Potomac River Park. Dr. Cushman has associated with himself Mr. Henry A. Gardner, who will have charge of the division of paint technology, and Dr. N. Monroe Hopkins, who will be in charge of the division of electrical engineering and electrochemistry.

Mr. **I. A. Goodell,** well known among smelting men in the West, was in Denver last month, having completed the work on which he was engaged in Montana.

Mr. **H. A. Guess,** manager of the Federal Lead Company, Flat River, Mo., recently spent some time in Salt Lake City on professional business for the interests with which he is connected.

Mr. **George A. Guess** is now metallurgical superintendent for the Cerro de Pasco Mining Company, Cerro de Pasco, Peru.

Dr. **Leo F. Guttmann** and Mr. **Camillo J. Guttmann** announce that the practice of their father, the late Mr. Oscar Guttmann, is now being carried on by them under the name of Oscar Guttmann & Sons. Specialties of their consulting practice are the erection of explosive plants, acetone, sulphuric acid, nitric acid and acetic acid plants, the fixation of atmospheric nitrogen as nitric acid, nitrates and cyanamides, the design, erection, remodeling and supervision of chemical factories, etc.

Mr. **D. C. Jackling,** of Salt Lake City, who is in charge of the operations of several of the large Western copper propositions, has found it necessary to deny a persistent rumor to the effect that he would assume general managership of all of the Guggenheim Western interests.

Mr. **Robert Kann** has formed partnership with Mr. R. J. Wysor in order to extend the scope of his laboratory. Mr. Wysor has been connected for a number of years with the Carnegie Steel Company. The new firm is established at 62-64 Gold Street, New York City, and is prepared to do general metallurgical work and to take charge of weighing and sampling of shipments.

Senor **Ramon F. Lujan,** of Mexico, has been spending some time in Denver on business relating to zinc production in Mexico.

Mr. **Charles Mindeleff,** chief chemist for the A. S. & R. Company, Perth Amboy, N. J., has been making a tour of the Western United States Mints in a professional capacity.

The Colorado School of Mines has issued a biennial report on the condition of the institution. In the last report, 1908, the enrollment was 367 students, of whom 46 per cent were from outside the State. At present the enrollment is 331 students, of whom 53 per cent are registered from outside the State. Six years ago only 26 per cent of the students were from outside the State. Thirty-six States, the District of Columbia, Alaska and the Philippine Islands are represented in the student body, in addition to seven foreign countries. The total number of graduates, including 1909, is 580. The inventory of property shows a total value of $723,841, a gain of 11.6 per cent in the biennial period. The report contains an appendix on non-metallic minerals of Colorado, by J. C. Bailar.

Digest of Electrochemical U. S. Patents.

Prior to 1903.

Arranged according to subject matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

Electric Furnaces (Continued).

695,939, March 25, 1902, Clarence S. Loman, of Everett, Mass.

Resistance type, for the production of oxides of tin and lead. The furnace is a horizontal trough having its ends widened and deepened to receive vertical terminal plates of copper. The narrow intermediate portion of the trough is covered by a collecting chamber, through one side of which project tuyères. A flue or chamber for the oxidized products opens from its other side. For the manufacture of binoxide of tin the metal is preferably maintained at a temperature of about 1200° C. and air is suplied at a temperature of about 400° C. To produce putty powder—a mixture of the binoxide and protoxide of tin—the molten tin is maintained at about 1200° C., and cold air is introduced. Cold air is also used for the manufacture of lead oxides.

696,004, March 25, 1902, George D. Burton, of Boston, Mass.

Arc-in-liquid type, for the smelting of ores of gold, silver, lead, etc. The charge is distributed in receptacles molded from fireclay and railroad cinders and baked. These receptacles have perforated flat bottoms dipping into a solution contained in a tank. The level of the solution is varied by a displacing plunger. The positive electrode is a vertical lead plate in one end of the tank. Each receptacle contains a negative electrode, imbedded in the charge, being a disk of carbon on one of copper, carried by a swinging arm. The receptacles are covered. In operation an arc is established between the liquid and the charge in the receptacles. The molten products pass through the perforated bottoms and are chilled in the liquid. The volatile products escape through a pipe leading from the cover to a condenser. Baffles in the pipe retain solid products. For the treatment of ores containing nickel, iron, sulphur and arsenic the solution may consist of water, 30 gal.; salsoda, 10 lb.; and sodium chloride, 2 lb. For the treatment of lead or silver-galena ores the solution consists of water, 30 gal.; and sodium chloride, 10 lb. For the treatment of silver-galena ores an electrode disk of antimony may be substituted for that of carbon. Ores containing sulphur, antimony and arsenic are fluxed with chlorides of sodium and ammonium. It is stated that ores containing gold, silver and lead give separate globules of these three metals. Oxygen may be injected into the escaping sulphurous fumes.

697,810, April 15, 1902, Michael R. Conley, of New York.

Resistance type, for the reduction of iron ores and production of steel for ingots. The furnace is a vertical stack, having a constricted portion toward its bottom. Radial electrodes of mixed plumbago and clay project through the walls at its neck. Other radial electrodes pass through the walls of the crucible beneath, which has tap holes for metal and slag. The metal decarburized at the neck can be recarburized or treated with manganese in the bath below.

698,766, April 29, 1902, A. Voelker, of Ehrenfeld, Germany.

Combined arc and resistance type. Materials for making glass are forced upward through conduits, by screw-conveyors, into electric arcs. The fused product runs downward into a refining pot, wherein it is heated by a current passed directly through it, between carbon-plate electrodes situated behind perforated partitions. The current density is 5 amp per square millimeter of electrode surface. The refined product overflows into a third chamber beneath. The walls of the melting and refining chambers are of chamotte.

BOOK REVIEWS.

More Recent Cyanide Practice. Edited by **H. Foster Bain.** Second edition. 418 pages, 88 illus. Price, $2. San Francisco: Mining and Scientific Press.

This is an appendix to (not a second edition of) "Recent Cyanide Practice," and is a reprint of articles on this process which have appeared in the *Mining and Scientific Press* from 1907 to 1910. In the 77 articles here reprinted is to be found some of the most recent information on all phases of the process. While too much in detail for general reading or students' use, yet the collection will be of great value to every practical user of the cyanide process.

Van Nostrand's Chemical Annual. A handbook of useful data for analytical, manufacturing, and investigating chemists and chemical students. Edited by **John C. Olsen,** A.M., Ph.D., professor of Polytechnic Institute, Brooklyn. Second issue, 1909. 580 pages. Price, $2.50 net. New York: D. Van Nostrand Company.

The first edition of this "annual" appeared in 1907, the second around January, 1909. Thus the term "annual" is misleading. However, there is really no inherent reason why a book of this kind should be republished every year. In spite of the rapid advance of chemistry in many fields a very large part of the book must always remain unchanged in succeeding editions; and many of the minor changes, especially those caused by the yearly changes in the "international" atomic weights, would be too insignificant to induce American chemists to buy a new edition every year.

The book is decidedly useful and has deserved the favorable reception which has been accorded to the first edition. It is what it is intended to be, a convenient reference book of numerical data, and it is especially to be recommended that all tables and numerical data have been quoted from the original source wherever possible.

There are introductory tables of atomic weights, physical constants of elements, etc. (47 pages). Then follow tables for the calculation of volumetric analyses (30 pages) and gas analyses (18 pages), physical constants of inorganic compounds, organic compounds, alkaloids and essential oils, and melting points and composition of lead-tin-bismuth-cadmium alloys (280 pages), specific gravity tables (64 pages), equivalents of weights and measures (10 pages), thermochemical data (12 pages). All this is excellent and very useful, though one might wish to have the thermochemical part considerably enlarged.

The book is concluded by a review of chemical literature and new books. This is well prepared. But as the review covers only "the more important articles" and is therefore acknowledged to be incomplete and as this information is easily available elsewhere (for instance, in "Chemical Abstracts"), the editor may find it useful to drop this part of the book when he needs the space for more tables in later editions. As it is the size of the work is just within the limits of a handy handbook. To enlarge the volume would make it without doubt unhandy.

Besides the editor, Dr. J. C. Olsen, the list of contributors includes Dr. B. B. Boltwood, Dr. V. J. Chambers, Dr. C. A. F. Kahlbaum, Dr. C. H. Lips, Dr J. L. R. Morgan, Dr. E. E. Reid, Dr. A. Rogers, Mr. A. F. Seeker, Dr. A. Seidell and Mr. W. C. Whipple.